MINING

MINING

AN ELEMENTARY TREATISE ON THE GETTING OF MINERALS

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LATELY

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PREFACE

THIS treatise is prepared exclusively for the use of beginners and for those who wish to teach beginners, or as a handy reference-book for busy men whose excess of knowledge and experience causes them sometimes to forget their early lessons. Still, the author must appeal to the good-natured forbearance of the reader, on account of the many shortcomings and imperfections of this little work. He has found great difficulty in compressing into the space at his disposal even the briefest reference to many subjects which have to be noticed in an elementary treatise on Practical Mining. He has also found it hard, in the midst of his practical engagements, to find time for any literary work.

He would like, however, to assure the reader that he has not ventured into print without endeavouring, so far as it was possible to him by labour and study, to prepare himself for the task. In his thirty-three years' experience in practical mining, he has been officially connected with mines as consulting mining engineer, colliery manager, and surveyor in Yorkshire, Derbyshire, Nottinghamshire, Leicestershire, Warwickshire, South Wales, North Wales, and Staffordshire. He has also from time to time inspected mines in Lancashire, Durham, Northumberland, Devon, Anglesea, and Scotland; and has visited many mines abroad in France, Germany, Austria, and in the United States of America. He is not only acquainted with coal-mines, but has experience of mines of clay, stone, iron, lead, zinc, copper, tin, gold, silver, and slate. For the last twenty-three years he has been responsible manager of coal-mines, and for the last eighteen years instructor and professor at the Yorkshire College. He has had charge of various sinking operations, and has made the working drawings for most kinds of plant and machinery required at a colliery; and has had to fix the price of all kinds of labour at collieries. He has also worked as a miner both at coal-getting and sinking. This has enabled him the more highly to appreciate the labours and writings of other engineers, to whose instruction he owes most of the information contained in this book.

He has derived great assistance from the records of the proceedings of numerous Institutes of Mining Engineers, of the Geological Society of London, the Institutions of Civil Engineers and Mechanical Engineers, and other scientific societies, and from the books of many able writers. He has found it convenient, as a means of saving time and labour, to make use in many cases of the drawings contained in these proceedings and books, though a great part of the six hundred illustrations are from his own original drawings and photographs. He wishes to thank the authors of papers and of books who have kindly permitted him to make reductions of their drawings, or who have specially given him information or opinions, viz. Messrs. T. Forster Brown, A. L. Steavenson, Wm. Galloway, Reuben Smallman, E. Reumaux, Professor Oscar Hoppe, Dr. C. Le Neve Forster, and many others, some of whom are specially acknowledged in other pages.

His object in calling attention to the works of others, whose individual experience is, as regards some of them, much greater than his own, and who are specialists of specialists on the subjects on which they have been consulted for this little book, is simply to let the comparatively inexperienced reader who may look at these pages know that he has the benefit of some of the knowledge obtained by others with unstinted labour; and if the reader should think that on some subjects the information given is not as full and complete as he would like, let him reflect that any one of fifteen out of the twenty-four subjects to which a chapter is given in these pages would require for its complete and masterly treatment a book as large or nearly as large as this.

LEEDS, 1893.

PREFACE TO THIRD EDITION

THE author has taken the opportunity of another edition being called for, to make a few necessary corrections and additions.

In view of the importance of the Explosives in Coal Mines "Orders," he has added the latest of these as an appendix.

He is indebted to the Controller of Her Majesty's Stationery Office for permission to reprint the Statutory Rules and Orders, which form Appendix A of this edition.

LEEDS, 1899.

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INTRODUCTORY CHAPTER.

COAL-FIELDS OF THE UNITED KINGDOM,

THEIR CONTENTS, PROBABLE DURATION, AND COMPARISON WITH THE RESOURCES OF THE REST OF THE WORLD.

THE accompanying map, which has been prepared from the information contained in the report of the Royal Coal Commission (1871), the Geological Survey, and other sources, shows the position and extent of the coal-fields of the United Kingdom. Where the coal-fields are concealed by newer formations, their probable extent is shown by vertical shading. The broken shading indicates that there is a strong probability of coal being found, but that the precise conditions are unknown. The experience of the last thirty-three years has tended generally to confirm the conclusions of the geologists who prepared the Coal Commission map.

A distinction has been made between the various formations, they having been divided up as follows:—

1. Pre-carboniferous rocks (Old Red Sandstone, Devonian, Silurian, Cambrian, etc.).

2. Lower carboniferous rocks (Millstone Grits, Yoredale rocks, Carboniferous limestone series, etc.).

3. Coal measures, including the Ganister beds.

4. Secondary rocks (Cretaceous, Wealden series, Oolite, Lias, Keuper, Bunter, Magnesian Limestone, Permian Sandstone, Conglomerates and Marls).

5. Tertiary rocks.

6. Alluvium.

By means of this map, the student will be able to arrive at a knowledge of where coal is and is not likely to be found. Thus it will be seen that the North of Scotland has at the surface pre-carboniferous rocks (Silurian), and these being lower in the scale of deposition than the carboniferous system (see p. 9), coal will not be found.

The south-eastern counties of England, however, are covered by secondary rocks (Lias, Oolite, Cretaceous), so that there is a possibility of coal being found below these in some places, though at what depth and in what places can only be ascertained by borings.

Referring to the map of Ireland, it will be seen that it has a floor of lower carboniferous rocks (carboniferous limestone). It seems probable, therefore, that at one time it was covered with coal measures, which have been removed by denudation.

The following is a list of the chief coal-fields in the United Kingdom :—

1. **The North Midland Coal-field.**—This coal-field extends through the counties of Yorkshire, Nottinghamshire, and Derbyshire, from Leeds and Bradford on the north to a few miles south of Nottingham. The western boundary of the coal-field is the millstone grit. The measures dip to the east under the Permian formation, and a great eastwardly development is now going on. The eastern boundary of the coal-field has not been proved, but it is probable that the area of the concealed coal measures will at least be equal to the area of the exposed coal measures, thus making a total area of about 1900 square miles, and forming the largest coal-field in the United Kingdom. The most easterly collieries in Yorkshire are Micklefield and Fryston on the north, and Denaby and Cadeby Main and Shireoaks on the south. Recently, however, a boring was put down at South Carr, about ten miles south-east of Doncaster, which proved several important seams, and reached the Barnsley Bed at a depth of 1061 yards, with a thickness of over 8 feet, but divided into two beds by a parting of 36 feet of shale.

In Nottinghamshire considerable developments have taken place in a line from Worksop through Mansfield to Nottingham, the most easterly pits being Manton, Shireoaks, Creswell, Warsop, Sherwood, Bolsover new sinking, and Gedling.

The number of collieries in the Midland Coal-field at the present time is 684, and the persons employed number nearly one-quarter of the total coal-mining population of the United Kingdom.

2. **The Northern Coal-field** includes the counties of Northumberland and Durham. It extends from the river Coquet nearly down to the river Tees, a distance of about 50 miles, the width varying from 5 to 30 miles. It has a similar structure to the Midland Coal-field, the western boundary being the millstone grit. Owing to its being on the coast, however, there is not the same possibility of extension under the newer formations. The Durham portion of the coal-field has an eastwardly extension under the magnesian limestone, which, however, has been practically fully proved. At the present time pits are being sunk at

Castle Eden (about four miles north of West Hartlepool), also at Seaham Harbour and Easington. These sinkings have got through the magnesian limestone, a mining engineering work of some difficulty, owing to the heavily watered condition of the limestone, due to the fissures communicating with the sea.

The extension of the workings underneath the sea has been fixed by various authorities at distances of from 3 to 10 miles. Coal has been worked under the sea here since the year 1872 at the North Seaton colliery. The royalties are payable to the Crown, the method of working most in favour being the long-wall system, in which the whole of the coal is extracted, provided there is a cover of about 90 yards or more, headings being carried in advance so as to prove any faults. The number of collieries in the Northern Coal-field is 440.

3. **The South Wales Coal-field** is fully proved and developed, the coal measures outcropping round the whole field. Owing to the existence of a great anticlinal which traverses almost the whole length of the coal-field, the deepest seams of the basin have been brought within workable depth. The mountainous character of this coal-field has greatly facilitated its development, because the upper seams cropping out on the hillsides could be easily discovered and cheaply worked by means of adits. The south-west portion of the basin is covered by the sea, and, owing to the erosion of the sea at Carmarthen Bay, there is a detached portion of the coal-field in Pembrokeshire. Travelling from south and east to north and west, the coal seams change in character from bituminous to anthracitic. In the intermediate part the celebrated smokeless steam coal is found. The number of mines at work in the South Wales Coalfield is 603.

4. **The Lancashire and Cheshire Coal-field** is separated from the Midland Coal-field by the Pennine Chain anticlinal, and at one time the two fields were undoubtedly connected. On the north and east it is bounded by the millstone grit, but on the south and west the coal measures are covered by New Red Sandstone, and extend in these directions.

The question of how far the coal measures extend under Cheshire has not been settled. The depth to the coal measures, supposing they extend as far, has been estimated at several thousand feet under the New Red Plain of Cheshire. The number of mines in the Lancashire and Cheshire Coal-field is 457.

5. **The North Wales Coal-field** includes the coal-fields of Denbigh, Flint, and Anglesey. On the west the coal-fields of Flint and Denbigh are bounded by Lower Carboniferous rocks, but on the east the coal measures extend under the Permian and New Red formations. On the Flintshire coast the coal measures

dip under the estuary of the Dee, and extend to the Cheshire side, being worked at Neston. The number of mines in the North Wales Coal-field is 65.

6. **The North Staffordshire Coal-field** is bounded on the east by Lower Carboniferous rocks, but the coal measures extend to the west and south under Permian rocks. A great thickness of coal measures have been proved with valuable seams of clay and ironstone. In some portions of the coal-field 140 feet of coal have been found (neglecting seams under 2 feet). The number of mines in this coal-field is 101.

7. **The South Staffordshire Coal-field** extends from Cannock Chase on the north to the Clent Hills on the south. It is noted, owing to the fact that the famous South Staffordshire Thick Coal, 10 yards in thickness, is found over a considerable area of the southern part of the coal-field. It probably extends under the New Red Sandstone to Warwickshire on the east and Shropshire on the west, and pits have proved the coal on the east at Hamstead and Sandwell Park, and on the west at Lord Dudley's new pit. The number of collieries in this coal-field is 246 in South Staffordshire, and 45 in Worcester.

8. **The Leicestershire Coal-field** is situated to the south of the Trent. The exposed coal measures have an area of some 36 square miles, but the extension on the west and south has been proved under the newer formations. On the north it is bounded by Lower Carboniferous rocks, and on the east by Pre-carboniferous rocks. The number of mines in this coal-field is 38.

9. **The Warwickshire Coal-field.**—The exposed portion of this extends from near Tamworth on the north to Bedworth on the south, and has a proved extension under the Permian rocks to the west and south. The Kingsbury Colliery, which was sunk a few years ago, is situated on the northern edge of the Permian measures of the Warwickshire Coal-field, and a series of seams found there have been correlated by some authorities with the South Staffordshire Thick Coal. The most southerly colliery in this coal-field is Wyken Colliery, but the coal-field undoubtedly extends south of Coventry. The number of collieries is 32.

10. **The Forest of Dean Coal-field.**—This small coal-field, with an area of 34 square miles, is interesting as being a perfect coal basin. It is bounded on all sides by the Lower Carboniferous rocks, the seams dipping east and west and north and south to a central axis. The number of mines is about 43.

11. **The Bristol and Somerset Coal-field** extends from Cromhall on the north to Frome on the south, a distance of 26

miles, and from Bath on the east to Bristol on the west, a distance of 12 miles, and it includes a total area of about 238 square miles. A very large portion of this coal-field, amounting to about four-fifths of the whole, is covered by newer formations. The coal measures are divided into two series by the intervention of a great thickness of coal measure sandstone, known as the Pennant rock. A very large area of the lower coal measures is virgin ground, which may ultimately be reached by sinking through the Pennant rocks. The upper division of coal measures is largely worked. The number of mines, including an outlying coal-field at Nailsea, is about 30.

12. **Shrewsbury, Coalbrookdale, and Forest of Wyre Coal-fields.**—The two latter coal-fields have easterly extensions under the newer rocks, possibly connecting the Coalbrookdale coal-field with that of South Staffordshire; but this connection is broken by the great Symon fault or washout which interposes a breadth of barren ground. On the west they are bounded by Pre-carboniferous rocks. The number of mines in Shropshire is 82.

13. **The Cumberland or Whitehaven Coal-field** is situated on the coast of Cumberland, and the coal measures extend under the Irish Sea, the coal seams being worked under the sea at Whitehaven and other places. It is bounded on the east by lower coal measures, and on the north and south by Permian rocks, under which there are probable extensions of the coal-field. The number of mines is 43.

14. **The Ingleton Coal-field,** on the borders of Lancashire and Yorkshire, is a small detached area of only a few square miles, partly overlaid by newer formations.

SCOTLAND.

The principal Carboniferous strata of Scotland lie in a long strip stretching between the Firths of Forth and Clyde in a north-easterly and south-westerly direction for a length of about 70 miles, and a varying width of about 20 miles, this area being broken up into a number of basins. The coal measures have been divided up into the Upper Series, underlying the red sandstones, and the Lower Series, underlying the white sandstones of the millstone grit. Below the Lower or Carboniferous Limestone Series, the Calciferous Sandstones occur containing oil shales. There are large areas of the Lower Series yet to be proved. The number of collieries in Scotland is about 460, the deepest shaft being 550 yards.[1]

[1] Presidential address by I. S. Dixon, "Trans. I.M.E.," vol. xxiii., p. 359.

The chief coal-fields of Scotland are:—

15. **The Clyde Basin.**—This is the principal coal-bearing area in Scotland.

16. **Midlothian and Haddington.**

17. **Fifeshire.**—This contains the Dysart seam, attaining 21 feet in thickness.

18. **Clackmannan.**

19. **Ayrshire.**

20. **Sanquhar.**

21. **Lesmahagow.**

There is a possible extension of the coal measures under the Firth of Forth in the neighbourhood of the Fifeshire and Midlothian Coal-fields. The Canonbie Coal-field of Dumfriesshire, which consists of Lower Carboniferous seams, may also extend under the Permian rocks southward to Solway Firth. As regards the other areas, they are all surrounded by Lower Carboniferous and Pre-carboniferous rocks.

Coal is worked to a large extent in the lower carboniferous strata of Scotland.

IRELAND.

As already mentioned, the coal-bearing areas of Ireland are very small, and there is no possibility of any concealed areas of any large extent being found, as the rocks are chiefly lower carboniferous and pre-carboniferous. The number of coal-mines in Ireland is 26. The coal-fields are as follows:—

22. **Antrim.**

23. **Leitrim.**

24. **Leinster.**

25. **East Munster (Tipperary).**

26. **West Munster.**

Royal Commission on Coal Supplies.—The appointment of a Royal Coal Commission this year shows that there are in this country many people of influence who consider that the information at their disposal with regard to our coal supplies, and our position as affected by those supplies, is insufficient. Any inquiry into the coal question must necessarily involve so much detail, and so many connected subjects, that it is probable that it will be some time before the recently appointed Commission is able to give the country the benefit of those inquiries. In the mean time, it may not be altogether wasted time to consider some of the information obtained by the Coal Commission of 1870 in the light of further knowledge and experience obtained in the last 30 years. The question may be divided into the following heads:—

A. How much coal is there in the United Kingdom?
B. How much of this coal will be available for our future use?
C. How much coal has been got from our original resources?
D. How long is our supply likely to last?
E. What will be the effect of the partial exhaustion of our coal supplies upon our manufacturing and commercial position in the world, having regard to the coal-fields existing in other countries?

In dealing with our coal resources, the report of the Coal Commission of 1870 may be taken as the basis.

The Commission divided our coal supplies into the following parts:

A. The coal contained in the coal-fields not covered by newer formations.
B. The coal contained in the coal-fields covered by newer formations, and which was thoroughly proved.
C. The coal in extensions of the known coal-fields under newer formations, not actually proved by sinkings and borings, but the existence of which was regarded as most probable by the geologists of the geological survey, as represented by Professor Ramsay and Mr. Prestwich for England, Professor Geikie for Scotland, and Professor Jukes, succeeded by Professor Hull, for Ireland.

The Commission decided that coal below 4000 feet in depth was unworkable, and that it would not take any account of coal seams less than one foot in thickness. It also decided to make large deductions from the amount of coal supposed to be in the ground on account of faults, barriers, and waste in working, the amount of such deductions varying from about 25 per cent. to 47 per cent.

Criticisms of Royal Coal Commission.—Before giving the figures as ascertained by the Royal Coal Commission, it may be well to criticize them to some extent.

Temperature of Mines.—The probable heat of mines may be estimated by the following calculation: Taking the temperature of the earth at a depth of 50 feet as 50 degrees Fahr., then add 1 degree Fahr. for every additional 60 feet; thus an additional 3000 feet would give an increase of 50 degrees in the temperature, or a total temperature of 100 degrees.

At a depth of 4000 feet the increase in temperature would be 66 degrees, or a total temperature of 116 degrees. At a depth of 6000 feet the increased temperature would be 100 degrees, and the total temperature would be 150 degrees. At a depth of

10,000 feet the increased temperature would be 166 degrees, and the total temperature 216 degrees, or, say, the temperature of boiling water. It is not likely that there is much coal in this country two miles deep, but it is likely that coal may be found one mile in depth. At first sight it might seem impossible to work coal at these great depths owing to the temperature, but a different opinion may be formed when we consider that coal and the associated rocks are bad conductors of heat, and that a blast of cold air upon the surface will soon make that surface quite cool. In 1868 the author calculated that it would be possible to introduce a sufficient weight of cold air into a mine to enable miners to get the coal at a depth of even 10,000 feet. The limit fixed by the Commission of 4000 feet has been very nearly reached, if not actually reached, in Belgium, without any serious difficulty from the high temperature. The deepest workings in this country are those of the Pendleton Colliery, where a depth of 3474 feet has been attained. It does not appear, therefore, that the temperature of the rocks will be an insuperable difficulty in getting our deepest supplies of coal.

Pressure of Superincumbent Strata.—A more serious difficulty in deep mining seems to be the great weight of rocks overlying the seam of coal, and the difficulty of maintaining roadways. This difficulty has not hitherto been found insuperable in any depth yet reached by mines. It has been found considerable in mines 1000 feet in depth, and mines between 3000 and 4000 feet in depth have not found the difficulty much greater. It is considered by many geologists that the rocks found at a greater depth will be harder than those found at a less depth, and that therefore they will be able to carry the weight of strata above.

Working of Thin Seams.—No sufficient reason has been shown why any traceable seam of coal 3 inches in thickness should be left in the ground as unworkable, in those remote days when all our coals over 1 foot in thickness have been exhausted. It seems that the difficulty of getting a thin seam, the same as the difficulty of working a hot mine, or a mine where the pressure is very great, can be entirely overcome if only a sufficient price is paid for the work, and this price will probably not be prohibitory. At the present time the working of seams only 20 inches in thickness is quite common in this country, and occasionally seams only 12 inches in thickness are worked. In the neighbourhood of Leeds a seam of coal has been largely worked in districts where it does not average over 16 inches in thickness, and is often much thinner. There is an instance of a seam in Germany, into the workings of which the author has been, which is only 4 inches in thickness. This 4-inch seam is not a seam of coal, but of black shale, which contains copper and a small percentage of silver, but, nevertheless, it is a 4-inch seam. At the time when the author visited the mines they were working hundreds of acres a year, at a depth of 1200 feet, and the cost of getting this 4-inch seam was about 13*s.* a ton, including every charge. Since this seam was heavier than coal, and the tonnage consequently larger, it may be considered that the cost would be equal to the cost of working an 8-inch seam of coal, so that we know that at a depth of 1200 feet an 8-inch seam of coal can be worked for 13*s.* a ton, and it is therefore probable that a 4-inch seam could be got for 25*s.* a ton. This price is not at all a prohibitive price. In our colonies coal is largely used at a price of £2 and upwards a ton, and in this country, within the last two years, coal has been sold at the pit's mouth for 20*s.* a ton, and some coal was sold at that price in the year 1874.

In dealing with the coal question, we are not dealing with questions of to-day or to-morrow, but with what is likely to happen hundreds of years hence, and it is safe to prophesy that in 500 years' time from now, 30*s.* a ton at the pit's mouth will not be considered an altogether unreasonable price, and at this price even a 4-inch seam of coal could be brought from great depths.[1] But the line need not be drawn at 4 inches, and coal will be worked in this country when it costs £3 a ton to get, and many an old gob will be ransacked for the slack, inferior coal and pillars which have been left behind.

Waste in Working.—The amount of coal wasted in working is a less proportion now than it was 30 years ago, and 30 years hence the waste will be still less, and in 300 years' time the waste in working will not be 2 per cent., but it is probable that there will always be a considerable amount left in barriers. If barriers are not left to keep the water from going down to the deep mines, more coal will be consumed in pumping than would be left in the barriers. But the barriers need only be left near the outcrop, and it will not be necessary to leave large barriers between different properties in deep mines, and also in deep mines it is probable that the gob will get sufficiently tight to prevent the passage of water. In the future the barriers left will probably not exceed 5 per cent. of the total, and the waste in working will not exceed 5 per cent., so that an allowance of 10 per cent. off the total quantity is sufficient. By adding to the figures of the Royal Coal Commission the deep coal, the thin coal,

[1] Mr. John Gerrard, in giving evidence before the Royal Coal Commission of 1902, said that three cases had come to his knowledge of a 10-inch seam being worked independently.

and three-fourths of their allowance for waste in working, the estimated tonnage of coal available for our supply is greatly increased.

Unproved Coal: Geological Anticipations.—In some respects the anticipations of the geologists of the Coal Commission have been fulfilled. The boring at South Carr, in Lincolnshire, has proved the existence of the coal-field 12 miles east of Doncaster, the coal seams lying level, and there is no reason to suppose that they may not continue another 12 miles or more. Indeed, there are people who say that the coal-field underlies the whole of the county of Lincoln; but that is a guess. The boring between Newark and Lincoln is unsatisfactory. Some people say it proves that there is no coal there, others dispute that contention; but, in any case, that boring is outside the limits included in the calculations of the Coal Commission. The sinkings at Sandwell Park, and Hamstead, near Birmingham, have proved valuable extensions of the South Staffordshire coal-field. As far as can be seen from an examination of the map, none of the areas included by the Coal Commission in their calculations have been proved *not* to contain coal.

On the other hand, the boring near Dover has proved the existence of workable seams of coal in Kent, as was held probable by some of the geologists consulted by the Coal Commission, but no figure was included in their returns on account of this coal. Few, if any, people expect very large coal-fields to be found in the South of England; but after the success of the Dover boring it will be unreasonable not to make some allowance for coal to be discovered in that part of the country. The average allowance of the Commissioners for waste in working was apparently about 40 per cent., leaving 60 per cent. to work. If this waste is reduced to 10 per cent., there would remain 90 per cent. to work, and the resources, as given by the Commissioners, would be increased by 50 per cent. Then as to the thin coals. It is very difficult to estimate the amount of those, and, indeed, the estimate can only be in the nature of a probable guess; but it is reasonable to assume that for every foot of coal met with in the seams of 1 foot and upwards in thickness there will be between 1 and 2 inches of coal met with in seams under 12 inches in thickness; or, in other words, that the thin coal will be not less than 10 per cent. of the thick coal. Therefore 10 per cent. may be added for the thin coal. Then comes the question of the coal in the South of England, as proved by the Dover boring. This boring so strongly supports the arguments of Mr. Joseph Prestwich and others, that a substantial figure should be added for this coal, say 5 per cent.

Anticipated Tonnage of Coal.—The total anticipated tonnage of coal is 359 thousand millions. The following tables give details of this total :—

TABLE A.

SUMMARY, AS PER COAL COMMISSION.

	Millions of Tons.
Exposed coal-field	90,207
,, ,, over 4,000 feet deep	7,320
Covered ,, proved and probable	56,182
,, ,, ,, 4,000 to 6,000 feet deep	25,840
,, ,, ,, 6,000 to 10,000 ,, ...	15,302
Total	194,851

ADDITIONAL TONNAGE ADDED BY THE AUTHOR FOR REASONS ALREADY GIVEN.

50 per cent. for reduced waste in working	97,426
10 ,, for coals under 1 foot thick	29,227
Coal-fields of Dover and possible South of England	17,536
Possible coal-field of Lincolnshire	20,000
Total	359,040

COAL COMMISSION.

SUB-PERMIAN : PROVED AND PROBABLE.

	Square miles.	Not over 4,000 feet.	Square miles.	4,000 to 6,000 feet.	6,000 to 10,000 feet.
		Millions of tons.		Millions of tons.	Millions of tons.
Warwickshire	84	2,485			
Leicestershire	42	1,790			
Midland and Black Burton	903	23,033			
Ireland		27			
District between :—					
Warwickshire and South Stafford ...	116	3,400			
South Stafford and Shropshire ...	195	5,800	} 112	} 3,346	
South Stafford and Coalbrook Dale	200	4,580			
East of Denbighshire	50	2,489			
W. and S.W. of North Staffordshire ...	50	1,500	75	2,240	
Cheshire, West of Kerridge	9	62			
,, between Woodford and Denton	36	1,790			
Lancashire E. and W. of Manchester	30	350			
Eccles to Runcorn, etc.	130	3,883			
Wirrell, Mersey and North	216	3,000	108	1,500	
Vale of Eden	40	1,593			
Severn Valley	45	400			
Denbighshire and N. Staffordshire ...			340	11,850	11,850
South of Manchester and Stockport ...			208	6,904	3,452
Total	2,146	56,182	843	25,840	15,302

COAL COMMISSION.

Coal 1 Foot and Over. British Exposed Coal-fields.

Name of Coal-field.	Square miles.		Below 4,000 ft.
		Millions of tons.	Millions of tons.
South Wales	878	32,456	4,109
Forest of Dean	33	265	
Bristol	166	4,218	1,885
Warwickshire	34	458	
South Stafford, Coalbrook Dale, and Forest of Wyre	309	1,906	
Leicestershire	36	836	
North Wales	127	2,005	
Anglesey	8	5	
North Staffs.	85	3,825	1,000
Lancs. and Cheshire	624	5,546	90
Midland	962	18,172	234
Black Burton	14	70	
Northumberland and Durham	789	10,036	
Cumberland	90	405	
Scotland	695	9,843	
Ireland	396	155	
Totals	5,240	90,207	7,320
Total		97,527	

Money Value of Minerals.—It might be worth while to consider the significance of the above figures in relation to some calculations of value. A great deal has been heard about the enormous mineral wealth in different parts of the world, in Klondyke, in West Africa, in the diamond fields of Cape Colony, and in the gold-fields of the Transvaal, and an attempt has been made to dazzle us by statements as to the great value of the minerals in these places; but the value of minerals as they lie in the ground is one thing, and their value when placed upon the pit bank quite another. If the popular plan of valuing the minerals under the earth at their probable value when placed upon the bank is adopted, it is safe to say that, looking into the future, the coal of Great Britain will average in value, when it is placed upon the pit bank, £1 a ton, and thus it will be seen that the value of the coal alone in the United Kingdom is 359 thousand million pounds, and, pursuing the same line with regard to the other minerals, the ironstone, clay, slate, lead, copper, tin, stone, salt, lime, and gypsum, it is safe to add as much more.

The result is that the total value of British minerals is 718 thousand million pounds.

TABLE B.

ANNUAL COAL PRODUCTION OF THE WORLD: IN MILLION TONS.

Period.	Great Britain.	France.	Germany.	Belgium.	Austria.	United States.	Other Countries.	Total.
1800	10	0·8	0·3			0·2	0·2	11·5
1820	12	1·2	1·5	1·0		0·5	0·5	16·7
1840	30	3·3	3·4	3·9	0·4	1·8	2·0	44·8
1850	49	4·4	6·7	5·8	2·0	8·0	5·5	81·4
1860	82	8·3	16·7	9·6	3·5	15·2	7·0	142·3
1870	110	13·3	34·0	13·7	9·5	32·9	9·0	222·4
1880	147	19·4	59·1	16·9	16·1	70·5	11·0	340·0
1889	177	24·6	84·9	19·8	24·0	142·0	12·7	485·0
1899	223	31·0	136·0	22·0	37·0	230·0	44·0	(723·0 metric tons)

Duration of our Coal Mining Industry.—Assuming that the resources of coal in this country have been correctly stated, it follows that at the present rate of production our coal will last us for 1500 years. But it is not probable that we have yet reached our maximum output. If our production is increased 50 per cent. our coal would last over 1000 years. If our production is doubled it would last nearly 800 years. He would be a bold man who would venture to prophesy what our production of coal is likely to be in the future. But we might be guided by the following considerations:—

Competition of other Countries.—At the beginning of the 19th century, Great Britain supplied seven-eighths of the estimated coal production of the whole world. At the end of the 19th century Great Britain supplied three-tenths of the production of the whole world. In the hundred years the production of coal in Great Britain has increased 20-fold, but the production of other countries has increased 300-fold. A hundred years ago British coal supplied America and Asia. To-day it is true that British ships still carry coal to all parts of the world, in absolutely larger quantities than 100 years ago, but it is only in Europe and the Mediterranean that we send any relatively large part of our export, nine-tenths of our exports being confined to these nearer countries. If we sail Eastwards, as soon as we have passed the Cape we find ourselves in competition with the coal-fields of Natal, India, Borneo, and Australia. If we sail Westward, we are beaten out of the field by the coal-fields of Canada and the United States; and doubtless this tendency to reduce our proportionate export to distant countries will assert itself still more in the future. Most countries contain coal suitable for their own requirements, therefore whilst our exports in the near future are likely to increase, that increase will probably be chiefly to our European neighbours and other Mediterranean ports. Norway, Sweden, Denmark, and the northern half of Russia seem to have but little coal. Holland has hardly any. The coal-fields of Belgium and France will probably not last more than 100 to 200 years. Germany alone in Northern Europe has stores of coal comparable to ours. There is a great coal-field in Southern Russia, which will help to supply Eastern Europe.

Spain has considerable coal resources, but it is not probable that she will be a large exporter of coal, and outside of Germany, Europe and the Mediterranean will in the distant future be large importers of coal, either from England or America, or both. At the present time American coal exporters are considering the Mediterranean as one of their future markets, and if the price of English coal were maintained at the price of 1900 there is little doubt but that American coal would largely supply the Mediterranean. In venturing to make a forecast, we may say that when our annual coal production has reached 300,000,000 tons we shall be getting, on an average, coal that is more expensive to get than the seams we are now working, and that at that period, compared with North America, our cost of working will be relatively higher than it is now, and that therefore American competition in the Mediterranean will be an important factor, and that, indeed, we may expect to see American coal largely supplying all countries that import their coal from a great distance. Australia will probably be also a large coal exporter. Therefore the time may arrive, very likely in the lifetime of many men now living, when the production of coal in Great Britain will become stationary, and that therefore we may look forward to the duration of our coal-fields for at least 1000 years.

Coal-fields of other Countries.—It is not possible to say exactly what are the coal resources of the world. It is probable that the coal-fields of Great Britain are in extent only about 1 per cent. of the area of the coal-fields in the world, but the tonnage of coal stored in the British coal-fields is probably a much larger percentage, and our coal resources are very likely from 3 to 5 per cent. of the coal resources of the world. But in considering the effect of these large coal areas upon the production of English mines, we must remember that the population of the United Kingdom is only about 3 per cent. of the population of the world, and that although our population is pre-eminently a coal consuming population, yet still the inhabitants of other countries are rapidly learning the use of coal. The production of Japan is now about two-thirds of what was the production of England 100 years ago, and if the Chinese should bethink themselves to make use of the power of their coal, it is not unreasonable to suggest that in the course of, say, 150 years they may produce coal in quantities proportionate to their coal reserves. With regard to North America, although their production per head of population is as yet only two-thirds of the British production, yet it is increasing with such rapidity that it is very likely that the production of coal in the United States per head will soon equal that of the United Kingdom per head of population; and when we consider the relative areas of the two countries, and their climate and natural fertility, there is no reason why the population of North America should not increase to such an extent as to reduce its reserves of coal as quickly as the reduction is proceeding in Britain. Of course, this will not come about all at once; 100 years will see a great change.

TABLE C.

STATISTICS OF THE WORLD.

	Coal-field. Areas proved or probable.	Present population probable.	Area of land.
	Sq. miles.		Sq. miles.
United Kingdom, say	9,000	42,000,000	121,000
Rest of Europe ,, ...	20,000	318,000,000	3,379,000
North America ,, ...	250,000	100,000,000	8,400,000
South ,, a guess	50,000	30,000,000	6,600,000
China ,, ...	200,000	800,000,000	17,000,000
Rest of Asia ,, ...	200,000		
Australasia and Oceana, a guess...	150,000	10,000,000	4,000,000
Africa ,, ...	100,000	200,000,000	11,500,000
Total	979,000	1,500,000,000	51,000,000

American Coal-fields and British.—In considering whether or not American coal-fields will last longer than the coal-fields of Britain, we must bear in mind that practically we start on the race for exhaustion on a level. Although coal has been worked in Britain for hundreds of years, it was not until the 19th century that any considerable tonnage was raised, and at the present moment we have only got 9,000 millions, or $2\frac{1}{2}$ per cent. of our original quantity. We have 97 per cent. still intact.

Economies in the Use of Coal.—As our coal-mines get deeper, and we work thinner seams more generally, the cost will increase, and the advantage of using fuel with the utmost economy will become increasingly evident.

At the present time economy in the use of coal is practised on steamships. There is greater advantage from economy on board a steamship than anywhere else, because for every ton of coal that is saved another ton of freight may be carried, and in some cases the freight is worth as much as the coal, so that twice the price of the fuel is saved. This great economy justifies a correspondingly large outlay in the purchase of new machinery of the best kind. It happens in a great many cases with engines on land that the expense necessary to secure the maximum economy is not incurred because the interest on the capital outlay required would absorb all or most of the profit to be gained by the economy, and thus we find that on land, engines commonly use from twice up to ten times the amount of fuel that would be necessary if the same economy was practised that is common in marine steam engines. In many chemical and metallurgical processes the consumption of fuel could be greatly reduced by a sufficient expenditure of capital. The same remark applies to brick works and pottery works.

For domestic purposes coal in this country is generally burnt in the most wasteful manner possible. The reasons for this waste are twofold. Firstly, the national taste for an open fireplace, which is liked because it is cheerful and because it is wholesome, being an effective agent in the ventilation of a room. In the case, however, of poor people the open fireplace is maintained because they for the most part do not live in their own houses, and cannot afford the outlay of capital necessary for the substitution of economical stoves for the open fireplace. Exactly how much coal is wasted in the open fireplace can hardly be stated; it varies with every house; but it is probable that with the most economical arrangements reasonably practicable one ton of coal would do as much work for domestic purposes as is now done by six.

The gas-engine is now competing with the steam-engine on a large scale, and it is probable that a considerable economy of fuel will be effected by its means. 1 lb. of coal per hour will give one horsepower in a gas-engine using producer gas.

Ultimate Economy.—Having regard only to methods of economy of which we have knowledge, it is not too much to say that on the average, taking every kind of use to which coal is put, its use can be so economized that one ton of coal can be made to do as much work as three tons do now. This economy will come about naturally as the price of coal rises, and this economical use of fuel will to a very large extent mitigate any injury we might otherwise suffer from an increased price of fuel. If, therefore, we assume an increased production from our mines of 50 per cent., and the increased efficiency of fuel of 3 to 1, we have a useful work done by our coal-mines of 4½ times their present useful work, lasting for a period of over 1000 years from now, and enriching a population 4½ times as great as now enjoy its use.

This population, of course, will not be confined to our shores, because our export of coal is increasing at a greater rate than our production. Forty years ago our production of coal in one year was 82 million tons, and of that quantity we exported about 7 million tons. In 1900 our production of coal was 225 million tons, and our export of coal was 44 million tons, so that our export has increased from 9½ per cent. of 40 years ago to the 20 per cent. of to-day, and I should not be surprised if our total exports increase till they become from 25 per cent. to 30 per cent. of our total production. Nevertheless, we shall keep at home coal enough to support in these islands a population of 150 millions.

Of course, it does not follow that 150 million people will want to live in these islands. The world is wide, and there is very comfortable room in it for twenty times its present population, without any undue crowding. With the present known means of agriculture, the earth will supply these people with everything necessary for food and clothing.

At the present time there are very large areas of the world with a very scanty population. Take the case, for example, of the United States of America, with a vigorous and energetic population. If you go to that country you will find that the cultivated land is in narrow streaks across the territory. Canada, Mexico, and South America are almost deserted; Siberia and Central Asia have also but little population.

Substitutes for Coal.—It is often suggested, by those who have not given very close attention to the subject, that some substitute will be found for coal—some new substance or some new force discovered that will do the work that is at present done by coal. The matter has been very carefully considered by some of the ablest chemists, physicists, and engineers in the country, and whilst it is possible to suggest sources of power that might do the work of coal when there is no longer coal, it is not possible to suggest any more economical source of power.

Gas Wells and Oil Wells.—Natural gas is found in some parts of the United States and elsewhere (a little in England, in the county of Sussex), but it is not likely that the weight of natural gas in the earth is sufficient to supply a deficiency of coal for any great length of time, and the same remark applies to oil, the production of which, though large in one sense, is very small as compared with the production of coal.

Peat is a good substitute for coal for many purposes, but the weight of dry peat in the world is not sufficient to affect this question materially.

Electricity.—Occasionally one hears it said that the work might be done by electricity; but we have no means of getting hold of electricity except by the power produced by coal or some other well-known source of power. Electrical machines and conductors are only one device out of many known to the engineer for transmitting power from one place to another place.

Chemical.—Other people suggest that the chemists will find out some substitute for coal; but the chemists have analyzed all the rocks of which the earth is made, and they know that there is no substitute for coal. There are, indeed, some bituminous shales and asphalts which will burn, and which, no doubt, will be used when better fuel is scarce; but these are not of great importance, and therefore will not materially prolong the duration of the coal-consuming period.

Tides.—With these exceptions, all the rocks of which the earth is made have, so to speak, been already burnt, and, so far as the production of heat or power is concerned, are absolutely dead. Other people point to the water-power contained in the tides of the sea, but if any engineer will take the trouble to calculate the cost of making the reservoir or dock in which to impound the tide at high water, and the value of the power that might be got from the water as it ran into and out of the reservoir or dock, he will find that, with the exception, perhaps, of a comparatively few cases, it will not pay to try to obtain power from the tides until coal costs from £2 to £4 a ton.

Waterfalls and Rapids.—Another source of water-power is in the waterfalls and rapids on our rivers. These, of course, are well-known to everybody, and in this country most of our rivers are already utilized. In Scotland, Wales, and Ireland there still remain some rivers which are not yet marked off with dams every few miles, and it is probable that at some future day all this power will be utilized. But the total of it will be but a trifling amount compared with our coal power. Here and there great waterfalls, such as Niagara and those on the Zambesi, Congo, Nile, and other rivers, endow certain localities with a supply of power, but this cannot be counted upon for much in comparison with the power of our coal-mines. The horse-power of Niagara is about 4,000,000 h.-p., and is equal to the power of 48,000 tons of coal a day, used in the most economical manner now known—that is to say, it is equal to the production of 24 first-class collieries, viz. 14 million tons a year. In Canada there are many other falls and rapids capable of giving enormous powers, and doubtless in the distant future the river water-power of the world will be a great source of manufacturing energy, but only a small fraction of the present coal-power, and that will not be in the British Isles.

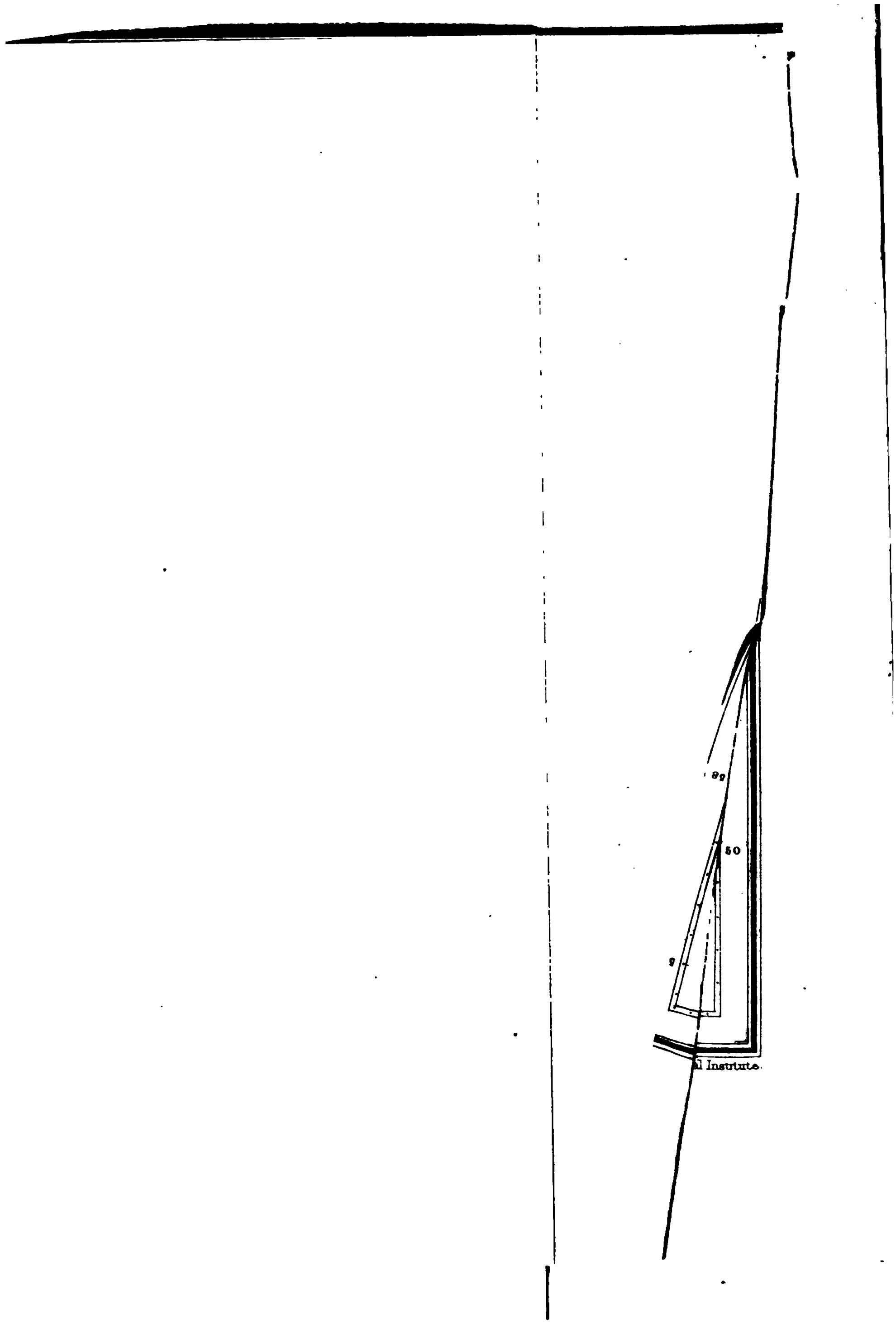
Wind-power.—The wind in this country is sufficiently powerful to do all the work which is now done by coal. It is not used much, because the use of coal is cheaper and more convenient, but when coal is scarce, then the power of the wind may be used.

Sun-power.—Power can be obtained by concentrating the rays of the sun upon a steam boiler. This has already been done experimentally. It was tried by Ericsson many years ago, and more lately larger apparatus has been erected in California.[1] A mirror 36 feet in diameter, containing 1000 square feet of surface, concentrates the sunlight upon a boiler, and gives 10 h.-p. At this rate, an acre might contain mirrors sufficient for 330 h.-p., or, say, 200,000 h.-p. per square mile. It is evident, therefore, that in the tropics and sub-tropics the sunshine alone will supply sufficient power for all the manufactures of the world.

In conclusion, the results of this inquiry may be summarized as follows:—

1. We have coal enough to last us for over 1000 years.
2. The rest of the world also has coal enough for its requirements for that period.
3. Coal in the future will be used with much greater economy than at present.
4. The industrial forces and population of this country may be increased fourfold and maintained with our coal-power for 1000 years.
5. The rest of the world also may share in the like increase.
6. The best probable substitute for coal when our coal is exhausted in this country, and in all temperate countries, is the power of the wind, and that will be sufficient, but power will have to be economized. In the tropics, sun-power will suffice.

[1] *Cassier's Magazine*, August, 1903.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MINING.

CHAPTER I.

GEOLOGY.

THE term "mining" is often employed to describe all engineering works below the surface of the earth, but in this treatise the term will only include those underground operations which are made in the search for, or the extraction of, minerals. For the due understanding of mining it is necessary to know something about the structure of the earth.

Shape of the Earth.—The earth is like a cricket-ball in shape, except that it is somewhat flattened at two opposite parts, called poles. Midway between the poles is the equator. The diameter at the equator is 26½ miles greater than the diameter at the poles; the average diameter is 7912½ miles.

The diameter at the poles is thus only shorter than that at the equator by $\frac{1}{300}$ of the maximum diameter. If a section of the earth is represented by a circle 3 inches in diameter (see Fig. 1[1]), the difference between the two diameters could hardly be shown without a magnifying-glass. Therefore, when using an ordinary pencil or drawing-pen to represent a section of the earth on such a small scale, it is best to treat the earth as if it were truly spherical, and a section of it as if it were a true circle (see Fig. 1), and so, also, it may be regarded for all the purposes of practical geology.

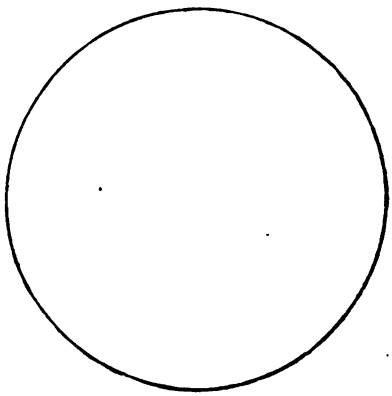


FIG. 1.—Section of the earth.

[1] This figure is less than 3 inches in diameter.

The surface of the earth is rough, the greater part being raised into hills (sometimes mountains), or depressed or eroded into valleys. The deeper valleys are generally filled with water, forming lakes, seas, and oceans. The highest mountain-tops in the world do not exceed 6 miles in height above the sea-level, and the depth of the deepest valley in the ocean probably slightly exceeds 6 miles; so that the height from the topmost ridge to the deepest furrow in the earth's surface is about 12 miles, or about $\frac{1}{660}$ part of the diameter of the earth. If a section of the earth is drawn 3 inches in diameter, as in Fig. 1, a fine pencil line will be more than $\frac{1}{660}$ of the diameter; and in the thickness of such a line will be included the highest mountain summits, and the deepest valleys of the Pacific Ocean. To represent the rough surface of the earth on such a small scale, it is necessary to draw a fine line, as in Fig. 1. The bottom of the deepest mines and bore-holes on the earth are not more than 1 mile below the sea-level (*i.e.* surface of the ocean), and on a section of the earth 3 inches in diameter the thickness of the finest line that a fine pencil can draw represents a greater thickness of the earth's solid crust than has been seen or touched by human eye or instrument, including in this thickness the depth from the highest mountain-top, not only to the bottom of the deepest mine, but 5 miles deeper still to the profoundest valley of the ocean.

Astronomers and mariners have taught us that the earth is round; but to others the earth seems flat, except for hills and valleys. If, instead of drawing a section of the earth upon a small scale, we were to draw it out to a natural scale or real size, we could not, of course, find room on the earth for a drawing-board of sufficient size to take it all in, and we should have to content ourselves with delineating a very small part of the earth. To draw a section of the earth real size by purely mechanical means, it would be necessary to use a compass with a limb equal in length to half the diameter of the earth, that is, nearly 4000 miles, but by trigonometrical calculations the use of the compasses is unnecessary, and a section of part of the earth could be drawn to the right curve approximately by means of a scale and straight-edge. If a line is drawn perfectly straight for 1 mile—say from the point A to the point B (Fig. 2)—and another line is drawn from the same point to represent the curvature of the earth's surface, this second line will be 8 inches from the line A B at the point B. But if,

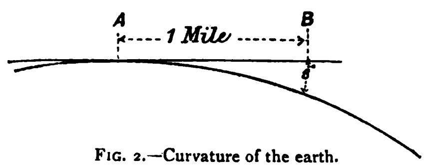


FIG. 2.—Curvature of the earth.

instead of a line 1 mile long, we take one only 10 feet long, and draw two lines, one perfectly straight and as thin as the finest steel point could scratch, from the point C to the point D (Fig. 3), and the other line to represent the curve of the earth from the same point C in the direction D, then, if a magnifying-glass were used, the two lines would appear to be identical; and it will be seen that, so far as the curvature of the earth is concerned, the surface of the earth is absolutely flat for all the purposes and considerations of the miner. It is well that this should be borne in mind, because it is a not uncommon error with people, who have looked at a section of the globe drawn to a small scale, to imagine that, by the curvature of the earth the rocks are formed into a natural arch which may cause them to bridge over large excavations. A little trigonometrical calculation, however, suffices to correct any such mistake.

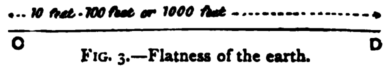


FIG. 3.—Flatness of the earth.

Composition of the Earth.—Looking at Fig. 1, the researches of the geologist reveal little or nothing as to that part of the earth which is within the thin line of the circumference. It is to the astronomer, or physicist instructed by him (paradoxical as it might appear to the unlearned), that we must appeal for information; and from him we learn that the weight of the globe is between five and six times that of an equal-sized body of water, or that the specific gravity is say $5\frac{1}{2}$, water being 1. We also learn that the earth is solid for a very great depth, probably for hundreds of miles; because, if the earth were fluid to within a short distance of the surface, it would be subject to tides as the sea is subject to them, and the movement of these subterranean tides would raise and lower the surface twice every day; but no such movement has been observed. In Fig. 4 the earth is shown in section. *a*, *b*, and *c* represent respectively depths of about 140, 280, and 790 miles; it would seem probable that, if the interior of the earth were

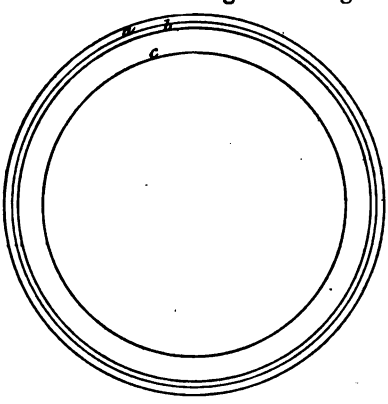


FIG 4.—Crust of the earth.

fluid, unless the solid crust reached nearly down to the depth *c*, it would daily yield to the influence of internal tides.

The average weight of the earth is say 5½ times that of water; but the rocks, as found on the surface and in the deepest mines, average about 2½ times the weight of water; so that the rocks which have not been seen must exceed the average weight.

The heaviest iron ore is 5·3; pure iron, 8·14; cast iron, 7·2; copper, 8·6; lead, 11·36; silver, 10·47; gold, 18·4; platinum, 21·53; aluminium, 2·56. It follows, therefore, that the rocks of the interior of the earth are denser than those at the surface, or else there must be a large proportion of the heavier metals to make up the required weight. But a little calculation will show that the substances inside the earth are subjected to intense pressure, which would tend to increase their density. The weight of the rocks on the surface is equal to at least a pressure of 1 lb. per square inch for every foot in depth; thus at 1000 feet depth the pressure is 1000 lbs. per square inch or more, at 1 mile depth, 5000 lbs. per square inch, and at 100 miles depth, 500,000 lbs. per square inch or more; but the hardest steel, if tested in small cubes, is crushed by this pressure; therefore it would seem probable that the materials of the earth at this depth must be more compact than they are at the surface, though by chemical analysis they might appear the same substance. But the amount of increase in density is an open question.

Heat of the Earth.—Numerous observations have proved that the temperature of the earth is greater inside than the average surface temperature. At a depth of 50 feet in England the temperature of the earth is 50° Fahr., and 60 feet deeper it is 51°, and so on. For every increase of 60 feet the temperature increases 1°, so that at a depth of 1250 feet the temperature is 70°, and it is 80° at 1850 feet depth. In some districts the rate of increase of temperature is much greater, and in others much less.

Hot springs of water, running for thousands of years, prove that in some localities there is great heat below, which is undiminished by the continual flow of water through these natural boilers.

Volcanoes prove that in some localities the internal heat is so intense as to melt very refractory materials. Geological investigation proves that volcanoes have existed in most parts of the earth.

Geology a Modern Science.—The science of geology in some respects is very ancient, and we inherit the benefit of the researches of practical miners and scientific men who were acquainted with the structure of the rocks before the days of Job; but, in its modern acceptation, it is for us comparatively new. It may be that, even in this regard, we have only rediscovered much that was known to learned men of the East in remote antiquity, the records of whose vast learning have been destroyed in desolating wars. But however that may be, it is within the last hundred years that most of the geological maps of the earth now existing have been made, and our knowledge of the earth's geological history (ponderous as are the volumes that contain it) requires many more generations of workers for its completion.

Geological History.—It may, however, facilitate some geological conception if we imagine that, at the beginning of geological history, as recorded in the stratified rocks, the surface of the earth was formed of hills and valleys, of dry land and sea, and that the surface rocks of that time were aggregations of mineral matter of the kinds which are frequently called igneous (or fire-made) rocks, perhaps similar to basalt, greenstone, granite (some kinds), trachyte, obsidian, porphyry, amygdaloids, volcanic ash, pumice, together with, it may be, such crystalline foliated rocks as are known by the names of schist, gneiss, etc.; and in some forms there must have been lime, iron, alumina, carbon, phosphorus, nitrogen, oxygen, and all the metals, minerals, and gases required for the formation of the rocks which are now visible. Then a process of destruction would attack the solid surface. Rain, wind, sun, frost, glaciers, streams, rivers, waves of the sea, chemical action, animal and vegetable action, would wear down the hills, and the fragments would be carried by the streams into lakes and seas, and be there deposited in beds which would gradually reach a thickness of thousands of feet, and consolidate into sandstone, shale, etc. (see Fig. 5), with which might be mixed lime abstracted from the sea-water; separate beds and masses of limestone might also be deposited in the sea, also rocks of other materials, such as silicates, might form as a crystalline deposit in the sea owing to some change of temperature or chemical constitution. When first the seas were condensed upon the surface of the earth, they may have been boiling hot, and have contained minerals in solution subsequently deposited in rock formations. It is also conceivable that the first seas were of fresh water,

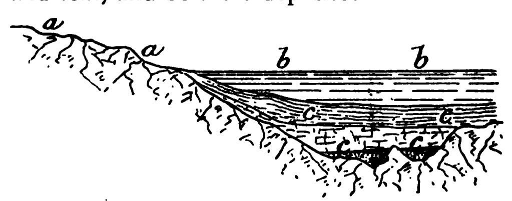


FIG. 5.—Original rocks and stratified rocks. *a*, original surface of the earth; *b*, surface of sea water; *c*, stratified rocks formed by *débris* from *a*.

becoming gradually impregnated with salt brought down by rivers.

Some parts of the earth are continually rising and other parts lowering, and many parts of the earth have been raised and lowered a great many times, first above and then below the sea-level.

The earliest sedimentary rocks were in course of ages lifted up above the sea-level, and the dry land lowered beneath it; then the new rocks would be attacked by the weather and carried down to the sea by the rivers and formed again into newer sedimentary rocks (see Fig. 6). These processes were repeated again and again through long ages of geological time, until the original surface of the earth entirely disappeared, and in its place was a covering consisting for the most part of sedimentary rocks, that is, rocks deposited as sediment at the bottom of lakes and seas. But mixed with these were great masses of igneous rock forced up through the newer rocks, and often flowing over and covering them (see Fig. 7). Many of the older sedimentary rocks have been so altered by pressure and heat that they have the appearance of igneous rocks, and are called metamorphic rocks; amongst these rocks are, perhaps, gneiss, serpentine, and quartzite. These processes of rock formation have continued up to the present time, and are still in active operation, though it is by some supposed that the destructive, and, as a natural consequence and concurrence, the constructive, agencies are less active now than in some earlier periods.

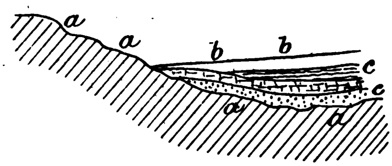


FIG. 6.—Older stratified rocks and new stratified rocks. *a*, old stratified rocks; *b*, surface of sea; *c*, stratified rocks formed from *débris* of *a*.

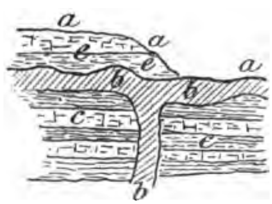


FIG. 7.—Intrusive igneous rock. *a*, surface of ground; *b*, intrusive igneous rock; *e*, newer strata; *c*, older strata.

A very remarkable and active agency in the formation of rocks is that of volcanoes (see Fig. 8). A volcano is a place where there is some fissure or opening leading from the surface to subterranean regions at a great depth, in a locality where rocks are, if not always, at least sometimes, in a molten condition; and during periods of eruption these are forced up the fissure in a huge stream of lava. Repeated eruptions often occur, covering large tracts of country.

Another form of eruption is that of the projection of very finely divided rock-forming material by a great blast of steam reaching to a height of miles, sometimes of 30 miles, forming a great cloud, which is carried a long way by the wind; the dust, falling to the ground, becomes consolidated into rock (called tuff).

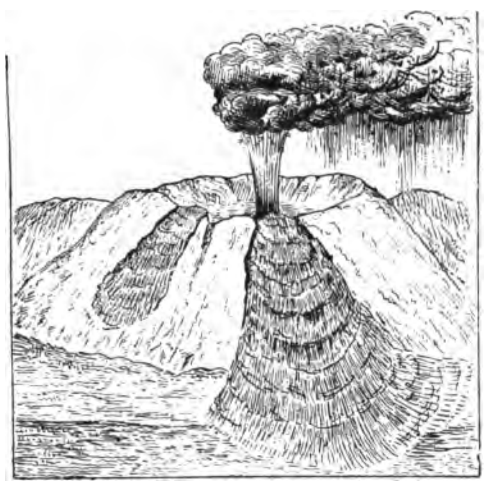


FIG. 8.—View of a volcano, showing stream of lava and eruption of ashes.

There are evidences of volcanic agency from the earliest known geological periods down to the present time.

In the earliest known sedimentary rocks no traces of life have been found. It may be that there were then no living things, either plants or animals, upon this globe, or that all traces have been obliterated by heat and pressure; but it is only reasonable to imagine that there was a time when life as we know it, that is, the life of plants and animals, first appeared upon the earth. The rocks in which no trace of life has been found are called archæan; they underlie all the other rocks and form their base and foundation. Animal and vegetable remains are found in all the later formations. In some cases the remains are merely casts; in others, a portion of the original plant or animal still remains. Thus vegetable remains are often accompanied by portions of the woody fibre, which is carbonized, or turned into coal; and animal remains, such as shells, are found, also bones and bony skin, though fossilized, that is, changed, except in the most recent deposits, by chemical reactions into stone. In many cases limestone rocks are almost entirely formed of animal remains (as in the case of coral islands), and masses of shells or other organisms. The casings of some plants and animals have served to form flint and flinty limestone, and animal remains have helped to form phosphatic limestone. At some periods large areas and great thicknesses of vegetable growths, such as trees and mosses, have been buried beneath deposits of mud, and this wood has been turned into coal by enormous pressure, perhaps accompanied by a high temperature. This is the origin of the coal-fields of the world. Fig. 9 shows in section some common alternations of strata in the coal measures.

FIG. 9.—Section of coal strata. *a*, bind; *b*, shale; *c*, rock; D, coal; *e*, seat, or fire-clay.

The other valuable minerals are sometimes found disseminated or aggregated in the stratified rocks, or as separate beds interstratified with the others; or else they are found as deposits in some cleft or cavity made long after the rocks themselves were formed.

One of the chief facts to be learnt by the student is that all the rocks on the explored surface or crust of the earth were not made at once, but have been made in regular sequence; and that those which were first made must underlie those which were made later; and that this rule is invariable, although, in cases where there have been great disturbances, some portions of the strata have been turned upside down.

The following is a list of the rocks found in Great Britain, arranged in order of date and superposition, the most recent and the uppermost being at the top of the column, and the earliest and deepest being at the bottom of the column:—

TABLE I.—BRITISH FORMATIONS.

	Recent.		
	Post-Pliocene.		
Cainozoic, or Tertiary.	Pliocene.		
	Miocene. (But slightly, if at all, represented in Great Britain by remaining strata.)		
	Eocene		Lower, Middle, and Upper.
Mesozoic, or Secondary.	Cretaceous		Chalk.
			Upper Greensand.
			Gault.
			Neocomian.
			Wealden.
	Oolite	Upper	Purbeck beds.
			Portland Oolite.
			Kimeridge Clay.
		Middle	Coral Rag.
			Oxford Clay.
		Lower	Cornbrash.
			Forest Marble.
			Bath or Great Oolite, and Stonesfield Slate.
			Inferior Oolite.
	Lias		Upper Lias.
			Middle Lias.
			Lower Lias.
	Trias		Rhaetic beds.
			Keuper (New Red Marl).
			Bunter (New Red Sandstone).

Palæozoic, or Primary.	Permian		Magnesian Limestone. Marls and Sandstones.
	Carboniferous		Coal Measures. Millstone Grit. Carboniferous Limestone.
	Devonian and Old Red Sandstone		Upper. Middle. Lower.
	Silurian	Upper	Ludlow Group. Wenlock Group. Upper Llandovery beds.
		Lower	Lower Llandovery beds. Bala and Caradoc beds. Llandeilo beds. Tremadoc Slates. Lingula Flags.
	Cambrian.		
	Torridonian		Sandstones.
	Archæan.		

In order to give some notion of the order and superposition of the stratified formations, an imaginary section (Fig. 10) has been sketched. All the strata were level or nearly level when deposited; the older rocks have been raised, denuded, and lowered several times, newer formations deposited on the denuded edges of older rocks, and finally there has been in recent ages a general upheaval on the western side of this imaginary section, raising the granite into a mountain and giving all the strata an easterly dip. The real dip would not be so steep as shown in the sketch, but it is generally convenient to exaggerate the depth and inclination in a geological section to save space.

The time required for the production of these rocks is too long for human conception.

To get some idea of geological time, it is only necessary to know that no traces of human life have been discovered on the earth in rocks older than the Post-Tertiary (Post-Pliocene) formations. These deposits have a thickness, where found in this country, averaging say 50 feet, though they occasionally occur in great masses, and at Boston in Lincolnshire are said to be 600 feet thick, and in some parts of Italy they attain a thickness of 1500 feet. These great thicknesses, however, are not equivalent to equal thicknesses of strata deposited as sediment at the bottom of a lake, because they occupy a comparatively small area, being moraines or *débris* scraped into a heap by glaciers. For comparison with other strata, it would probably be fair to consider the Post-Pliocene as being 50 feet in thickness, whereas the strata below have probably a thickness of 165,000 feet. But

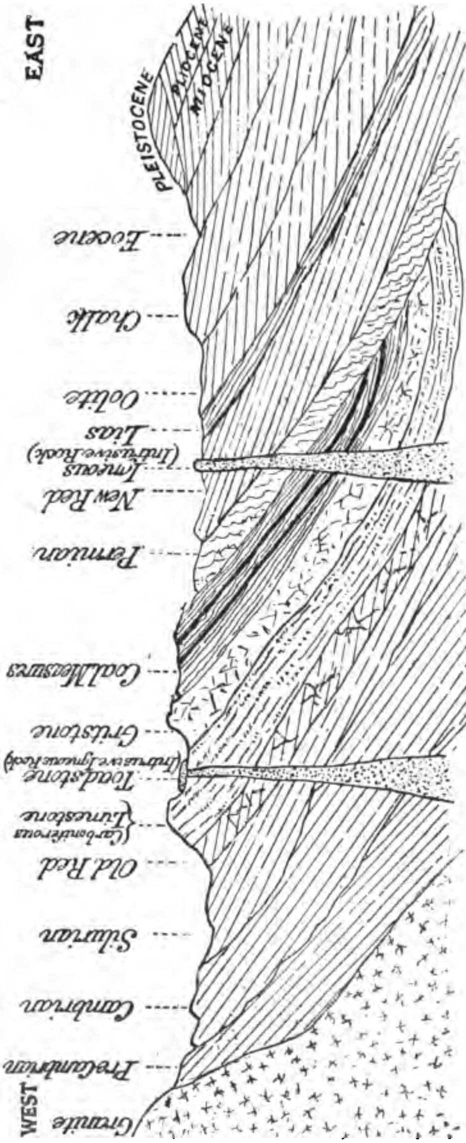


FIG. 10.—Ideal sketch, showing section of strata.

the length of time that has elapsed since human beings first came upon this earth is quite unknown, and those geologists who have most studied the subject hesitate to state their views in the figures of arithmetic, because of the absence of sufficient data for accurate statement; but for the purposes of elementary inquiry some geologists say that the earliest human remains date from a period 100,000 or 250,000 years ago, though others consider this estimate too high. And if a proportionate amount of time (as for the Post-Pliocene) were required for all the other strata

FIG. 11.—"The Gateway" in Colorado.

in the world, their formation would indicate a lapse of time of between 300 and 750 million years. But this estimate is certainly too high.

Another way of getting some idea of geological time is to watch the slow process by which the weather acts upon the hills at the present time, and try to calculate the period required to wear down a hill 500 feet high, and deposit it as sand and mud in the sea 100 miles distant. The hills and valleys of the present surface of the earth have been to a large extent sculptured by the gradual action of the weather.

This sculpturing action of the elements is shown in Fig. 11, which is a photograph taken by the author in Colorado, showing an opening, locally called "The Gateway," in a mass of rocks, gradually produced by the action of the weather. The distant mountain is Pike's Peak.

Fig. 12 is reduced from another photograph taken by the author in the same locality. The huge lump of stone rising above the bushes is locally called "The Dutchman," and shows how the softer and thinner beds are being gradually worn away by the weather.

FIG. 12.—"The Dutchman."

But the present hills are not the original hills of the earth. They are in many cases cut out of rocks made from the *débris* of older hills, and these, again, from still older hills, the processes having been repeated several times. Yet all our modern hills were ancient hills before the human race was seen upon their slopes.

Or, again, looking at the list of formations, notice how small a place is occupied by the coal measures. But the coal measures in some parts of Great Britain are believed to have a thickness of 10,000 feet, and contain in some places 100 feet of coal. Can we imagine the length of time required for the formation of 100 feet of coal?

The coal is made of vegetable matter or woody fibre that has grown on the spot. It is from two to three times as heavy as wood, so that at least 250 feet of solid wood are required to make 100 feet of coal. But as the wood grew and died it would not all remain; a great part of it would decay, and only some portions would be preserved. But if we assume that one-half was preserved, then it would require, to make 100 feet thick of coal, a growth of wood equal to a thickness of 500 feet.

It has been calculated that 9 feet of peat would be required to form 1 foot of coal. If the trees grew 1 foot high each year, and if for each diameter of 1 foot of trunk there was a diameter of 20 feet of ground, or 400 square feet for each square foot of trunk, then the time required would be $400 \times 500 = 200{,}000$ years for 100 feet of coal. And then there is the time required for 9900 feet of shale, sandstone, and fire-clay; if these beds were formed at the same rate as the coal, it would require 20,000,000 years to make the coal measures. But the coal measures are only one of three divisions of the Carboniferous formation, and the Carboniferous system is only one out of fourteen distinct systems.

Such calculations as the above are no guide at all as to the real duration of geological time, but serve only as examples of one way of getting some idea of possibilities in one particular locality. It is not probable that in any place the formation of rocks went on without interruption for a thickness of 10,000 feet, and any period of time guessed at by calculation might require to be multiplied by a large factor or divided by a large divisor to approach the real truth.

The different formations are recognized partly by their lithological characteristics, that is to say, by the quality and appearance of the rocks, and partly by the fossil remains of plant and animal life which they contain. Each formation has not got an entirely distinct set of fossils, but it has some distinctive fossils.

The recent rocks contain remains of animals very similar to many of those now existing on the globe, and, going backwards to the older rocks, the difference between the animals then existing and now is greater, until all forms of mammalia are lost; and it is supposed in the earliest rocks of all only the remains of comparatively elementary organisms exist. As regards the lithological characteristics of the various formations, the following observations may assist the student:—

The Cretaceous, or chalk rocks, are often conspicuous, as in the wolds of the East Riding of Yorkshire, in Lincolnshire, Hertfordshire, Wiltshire, or Sussex, where the chalk forms gently sloping hills, in which numerous quarries reveal the white limestone.

The Trias, or New Red Sandstone, generally gives a red colour to the ploughed fields, while the river-banks, railway cuttings, and quarries show sandstones and shales; sometimes thin beds of gypsum are interstratified.

The Millstone Grit is often seen in rough crags and precipices, as in the Pennine chain of Yorkshire and Derbyshire, and is itself a rough mixture of sand and pebbles consolidated into a hard and durable rock, which is largely used, not only for millstones, but for building.

The Mountain or Carboniferous limestone is seen in mountains of rounded contour, often cut by deep valleys with precipitous sides; it is extensively quarried for limestone.

The Silurian and Cambrian systems form the wildest mountains of North Wales, and from them come most of the roofing-slates used in this country.

The student soon learns the distinctive appearance of some of the strata of each formation.

With regard to the characteristic fossils of each system, the student may easily learn to recognize the Lias by the ammonites (see Fig. 13) and belemnites which abound; the Coal Measures,

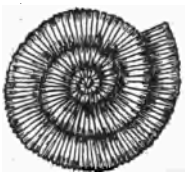


FIG. 13.—Ammonite

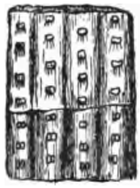


FIG. 14.—Sigillaria.

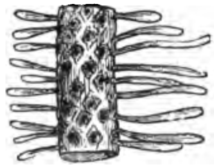


FIG. 15.—Stigmaria.

by the sigillaria and stigmaria (Figs. 14, 15, 16), lepidodendron (Fig. 17), alethopteris (Fig. 18), calamites (Fig. 19), goniatites (Fig. 20, etc.); the Mountain Limestone, by the encrinites, which in some places seem to make a large portion of the rock (see Fig. 21). The Silurian formation is noted by the development of

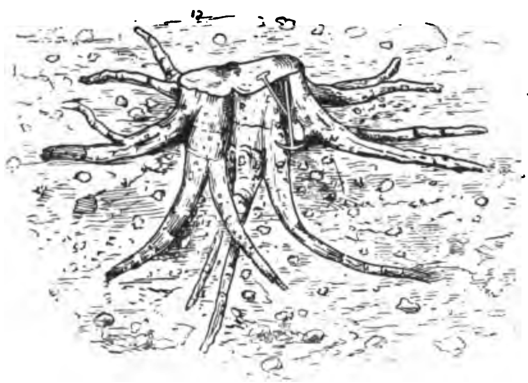


FIG. 16.—Stigmaria, from a photograph.

the crustaceans known as trilobites, of which a specimen is shown in Fig. 22.

Occurrence of Minerals.—All the coal in Great Britain (with slight exceptions[1]) occurs in the Carboniferous formation. It is always, without exception, in regular beds interstratified with

[1] These exceptions consist of a bed of coal about 2 feet thick in the Lower Oolite, on the hills between Whitby and Helmsley, in North-East Yorkshire; and some Carboniferous shale in Dorsetshire, in the same formation; the Brora coal, etc.

beds of fire-clay, shale, and sandstone, and sometimes limestone (see Fig. 9).

A bed of fire-clay underlies every seam of coal, and a bed of

FIG. 17.—Lepidodendron.

FIG. 18.—Alethopteris.

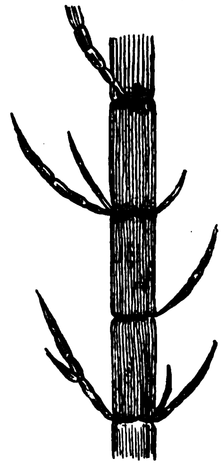


FIG. 19.—Calamites.

shale overlies every seam of coal. There are exceptions to these rules. Sometimes the fire-clay has a substitute, in the shape of a bed of nearly pure silica, called ganister, which is very hard, but

FIG. 20.—*Goniatites Listeri*, from "Baum Pot." Sketch enlarged from prepared section of fossil.

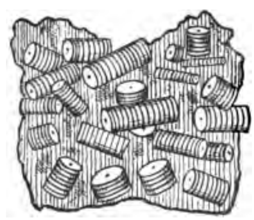


FIG. 21.—Weathered surface of crinoidal limestone.

FIG. 22.—Trilobite (*Ogygia Buchii*).

this is very rare; and sometimes, though seldom, a sandstone takes the place of the shale roof.

The seams of coal in Great Britain are remarkable for their regular thickness and the wonderful manner in which the characteristics of each particular seam are often maintained for many miles.

A seam of coal, say 4 feet thick, may sometimes be traced for 5 or 6 miles without a variation in thickness exceeding 6 inches, whilst the variations in quality will be hardly perceptible.

A seam of coal sometimes maintains its distinctive qualities for a distance of more than 40 miles.[1]

The natural tendency is towards variations, and coals thicken or thin, or split up into several seams (see Fig. 23) or are brought

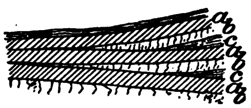
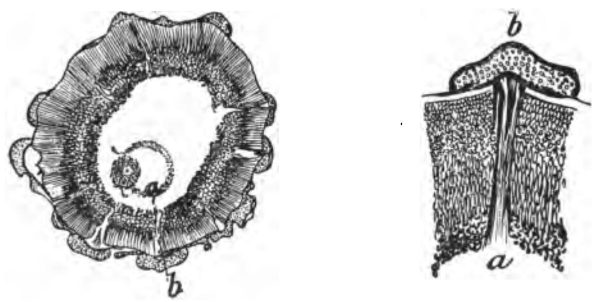


FIG. 23.—Three seams of coal uniting to form one. *a*, coal; *b*, fire-clay; *c*, intervening shales.

FIG. 24.—Sandstone merging into shale. *a*. sandstone; *b*, shale.

together into one thick seam (as in the Staffordshire thick coal, which is 30 feet thick in one great bed, with only thin dirt partings, and in a distance of about 10 miles intercalations of sandstone and shale are included, having an aggregate thickness of



FIGS. 25, 26.—*Lepidodendron selaginoidis* (transverse sections). Sketches enlarged from prepared section of fossil. Fig. 26, enlarged from Fig. 25 on line *a b*.

500 feet); they vary from clean to dirty and from anthracitic to bituminous.

In the same way the shales gradually merge into sandstones, and *vice versâ* (see Fig. 24). Soft fire-clays change into hard or indurated fire-clays. In one part of a coal-field two well-known seams may be separated by say 240 yards of strata, and within say 16 miles the intervening strata may be reduced to 180 yards. These changes are what must always be expected; it is only the regularity that at all surprises the geological student.

[1] Cannel coal is an exception to all the above statements.

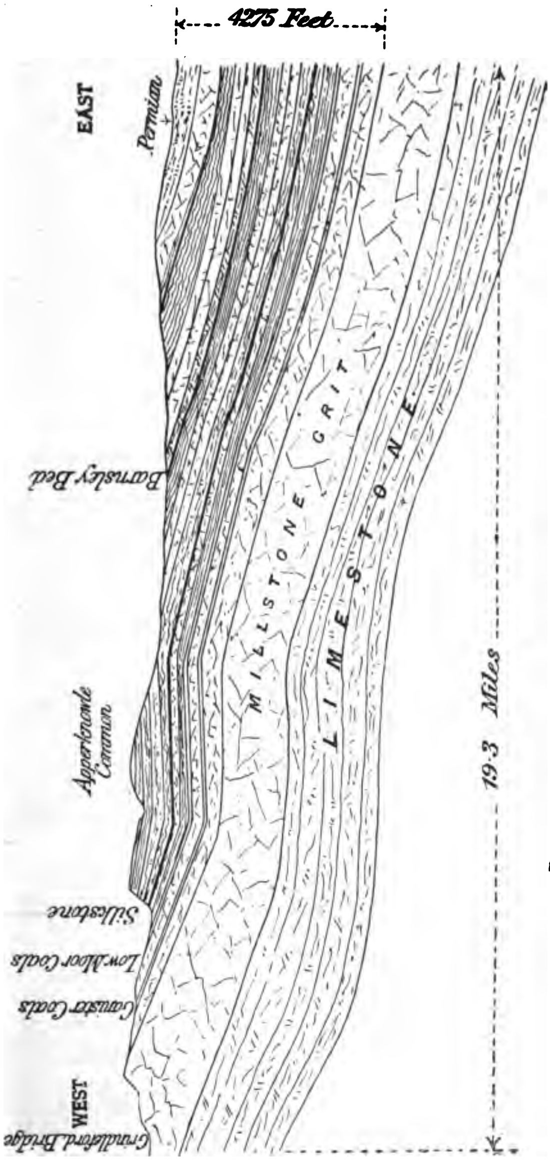


FIG. 27.—Section across Midland Coal-field, near Sheffield, from west to east.

In the Coal Measures the stumps of trees are often found erect, as if in the place where they had grown.

The origin of coal can be shown by grinding sections of it till

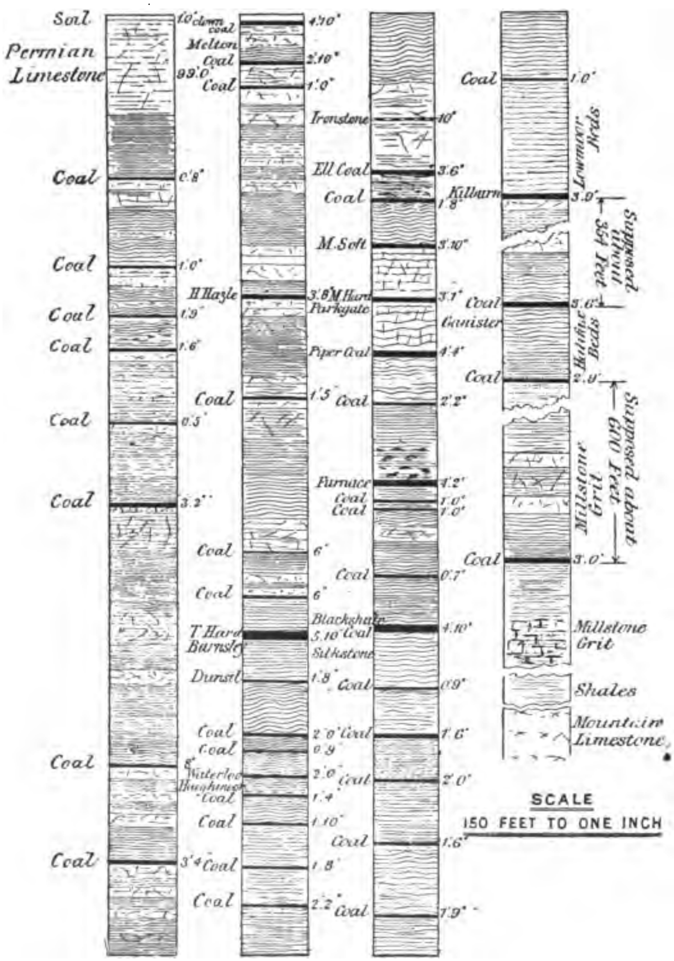


FIG. 28.—Vertical section of Derbyshire Coal-field between Chesterfield and Nottingham.

they are transparent, then, if a bright light is focussed on these sections and a double convex lens placed in front, the structure of the specimen may be displayed upon a white screen; and it is thus seen to bear the characteristics of wood and vegetable growth (see Figs. 25 and 26).

The following are some sections of coal-fields: Fig. 27, section across the Midland Coal-field; Fig. 28, vertical section of the Derbyshire Coal-field between Chesterfield and Nottingham.

The above sections were levelled and measured by the author for the Royal Commission in 1867–68.

The following list of minerals includes most of those known commercially to miners. Those printed in darker type are the most important as regards commercial value. Nearly all of them are got in British mines, though often only in very small proportions. But platinum, mercury, and the more valuable precious stones are not found in British mines. Some of the minerals and metals whose names are given below and in the following pages occur in a great variety of forms and chemical combinations, to each of which mineralogists have given a name. There are many hundreds of these, for the study of which the reader is referred to books on mineralogy and metallurgy.

TABLE II.—LIST OF FIFTY-EIGHT MINERALS.

E *denotes that they are found in Great Britain.*

Apatite (for phosphorus).	**Gold.** (**E.**)	**Petroleum.** (**E.**) (Only slight indications in Britain.)
Alum clay. (**E.**)	**Gypsum.** (**E.**)	**Phosphates of lime.** (**E.**)
Alum shale. (**E.**)	Graphite. (**E.**)	**Platinum.**
Aluminium ore (bauxite, cryolite, etc.).	**Iron ore.** (**E.**)	Potassium ore.
Antimony ore. (**E.**)	Iridium ore.	**Precious stones.**
Arsenic ore. (**E.**)	Jet. (**E.**)	**Salt.** (**E.**)
Asbestos.	Kaolin.	**Silver ore.** (**E.**)
Asphaltum.	**Lead ore.** (**E.**)	**Slates.** (**E.**)
Barytes. (**E.**)	**Lithium ore.** (**E.**)	Sodium nitrate.
Bismuth ore.	Lignite. (**E.**)	**Stone.** (**E.**)
Cadmium ore.	**Limestone.** (**E.**)	**Sulphur ore.** (**E.**)
Clays (Fire-clay). (**E.**)	Magnesium ore.	Strontium ore. (**E.**)
Coal. (**E.**)	**Manganese ore.** (**E.**)	Tellurium ore.
Cobalt ore. (**E.**)	Mica.	**Tin ore.** (**E.**)
Copper ore. (**E.**)	**Mercury ore.**	Tincal.
Flint. (**E.**)	**Mineral waters.** (**E.**)	Tungsten ore (wolfram). (**E.**)
Fluor-spar. (**E.**)	Molybdenum ore.	Uranium ore. (**E.**)
Ganister. (**E.**)	Ochre. (**E.**)	**Zinc ore.** (**E.**)
Gas (of no commercial value in Britain).	**Oil shale.** (**E.**)	
	Ozokerite.	
	Palladium ore.	

Of the above minerals many are found as beds of stratified deposit or disseminated in stratified rocks. Some of these are shown in the following table:—

TABLE III.

British and Foreign.

Alum clay or shale.
Bog-ore.
Fire-clay.
Coal.
Flint.
Ganister.
Gypsum.
Graphite.
Iron ore.
Lignite.
Jet.
Limestone.
Ochre and Umber.
Oil shale.
Petroleum (only slight indications in Britain).

British and Foreign.

Phosphate of lime.
Salt.
Slates.
Stone.
Stream deposits, containing gold, tin, and other metals.

Foreign only.

Asphalte (Val-de-Travers, Trinidad, etc).
Borax.
Gas.
Nitrate of soda.
Ozokerite.
Precious stones.

And others are found in veins,[1] that is, in cracks, crevices, clefts, caves, pipes, holes in rocks which were made and consolidated ages before the veins were formed.

These cracks have been caused by subterranean disturbances which have split the solid rocks often for many miles in a nearly straight line, sometimes raising one side (or lowering the other) and sometimes leaving an opening between the two sides of from a few inches up to more than 100 feet in width (see Fig. 29). Such a fracture of the strata, when accompanied by displacement, is commonly called a "fault." When a fault takes place in hard rock, an open crack is often left; but when the fault is in soft rocks, such as the ordinary coal measures, shales, and binds, no

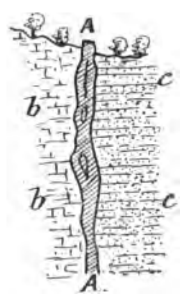


FIG. 29.—Vein in fault. A, vein; *b*, limestone rocks; *c*, sandstone.

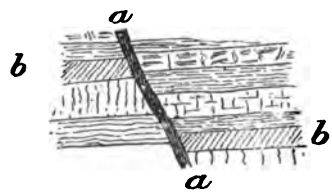


FIG. 30.—Fault in coal measures. *a*, fault filled with dirt; *b*, main coal.

[1] In South Wales and in those numerous countries abroad where Welsh miners are engaged, the word "vein" is applied to stratified seams of coal as well as to mineral veins.

open crack is left, the strata on each side being squeezed together (see Fig. 30). Sometimes a crack may be opened in rocks without raising one side above the other.

Holes are sometimes made in rocks (especially limestone rocks) by the action of water charged with acids, or by abrasion of gravel-charged streams. These holes are sometimes wide chasms, and sometimes narrow clefts, and sometimes like a pipe; in many cases the large chasms are connected by narrow clefts forming a chain miles in length. These cracks, caves, chasms, and pipes have been filled up with a mineral deposit containing vein-stuff (*i e.* valueless minerals) and ore (*i.e.* valuable mineral) in constantly varying proportions (see Figs. 31 to 33).

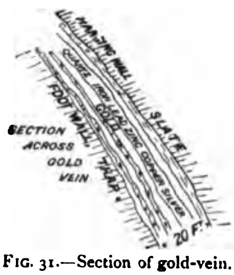


FIG. 31.—Section of gold-vein.

Where the veins are formed in cracks caused by violent

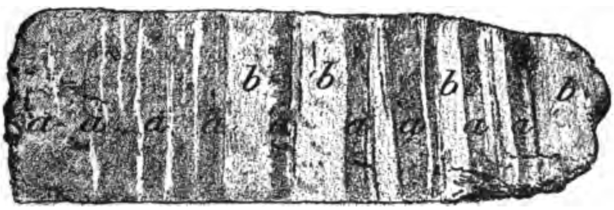


FIG. 32.—Silver-lead, Clausthal, Germany: from a photograph by the author. *a*, lead ore containing silver, as seen in a section of the vein: *b*, vein-stuff or spar (waste). The width of the vein here shown is 3 feet 9 inches.

disruption of hard rocks, they reach down to great depths, and in most cases the bottom of the vein has never yet been proved, and it may be beyond the reach of the miner (the deepest mine has not yet explored a vein for a vertical height of 5000 feet, though a bore-hole has reached nearly 6000 feet). There is no reason to suppose that, within such depths as are accessible to human beings, these veins will be either richer or poorer in valuable ore as the mines get deeper.

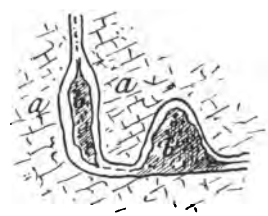


FIG. 33.—Water-worn cavity and mineral. *a*, limestone with water-worn cavities; *b*, mineral vein.

But as regards the veins which are found in water-made caves, the bottom of these is reached when the bottom of the cave is explored, and cannot go deeper than the stratum which is capable of being formed into caves by aqueous action.

Thus if iron ore (hæmatite) were found in water-made caves of the mountain (Carboniferous) limestone, it would be vain to explore the shales or sandstones underlying the limestone for a continuation of these deposits of ore.

The following list shows some of the minerals which are now mined in veins :—

TABLE IV.

British and Foreign.	Foreign only.
Antimony ore.	Apatite.
Arsenic ore.	Asbestos.
Barytes.	Bismuth ore.
Cobalt ore.	Cadmium ore
Copper ore.	Iridium ore.
Fluor-spar.	Lithium ore.
Gold.	Magnesium ore.
Graphite.	Mercury ore.
Iron ore.	Mica.
Lead ore.	Molybdenum ore.
Manganese ore.	Palladium ore.
Potassium ore.	Phosphate of lime.
Silver ore.	Platinum.
Sulphur ore (as pyrites).	Precious stones, diamonds, rubies, emeralds, turquoises, garnets, sapphires, opals, etc.
Strontium. ore.	Native sulphur. (Some geologists would not allow this as being got in veins of any kind.)
Tin ore.	Tellurium ore.
Tungsten ore.	
Uranium ore.	
Zinc ore.	

Sometimes minerals are found in volcanic craters and in pits or channels like huge shafts (see Fig. 34) formed by volcanic agency. Such, perhaps, are the Mount Morgan Gold-mine of Queensland, the Kimberley Diamond-mine of South Africa, the sulphur-mines of Sicily.

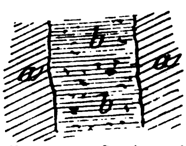


FIG. 34.—Section of volcanic shaft. *a*, country rock; *b*, volcanic shaft or vein, containing mineral.

The following table is interesting, as showing the tonnage and value of the minerals got annually in Great Britain. It has been compiled from the Government statistics:—

TABLE V.

Description of mineral raised in the United Kingdom.	1889.		1890.		1891.	
	Quantity.	Value at the mines.	Quantity.	Value at the mines.	Quantity.	Value at the mines.
	Tons.	£	Tons.	£	Tons.	£
Alum clay (Bauxite)	9,150	5,490	11,527	5,763	10,763	3,228
Alum shale	4,188	523	6,420	802	5,474	684
Antimony ore ...	(cwts.) 67	900	14	200	15¼	250
Arsenic	4,758	38,260	7,276	60,727	6,048	58,593
Arsenical pyrites ...	7,688	7,317	5,114	4,414	5,095	4,370
Barytes	24,849	38,238	25,353	29,684	26,876	32,120
Bog ore	14,002	7,001	14,512	7,256	16,075	8,037
Clays (excepting ordinary clay) ...	3,036,253	828,174	3,308,214	899,166	3,222,035	943,896
Coal	176,916,724	56,175,426	181,614,288	74,953,997	185,479,126	74,099,816
Cobalt and Nickel ore	155	968	84	260	Nil.	—
Copper ore	9,029	26,584	12,136	27,801	8,836	20,214
Copper precipitate ...	281	3,113	345	4,670	322	4,355
Fluor-spar	297	411	268	392	141	187
Gold ore	6,226	10,746	575	434	14,117	12,200
Gypsum	132,357	53,819	140,293	57,991	151,708	60,038
Iron ore	14,546,105	3,848,268	13,780,767	3,926,445	12,777,689	3,355,860
Iron pyrites	17,719	8,111	16,018	7,666	15,463	8,002
Jet	(lbs.) 618	124	(lbs.) 1,228	245	(lbs.) 766	153
Lead ore	48,465	429,647	45,651	406,164	43,859	356,783
Lignite	947	284	2,630	767	4,664	1,360
Manganese ore ...	8,852	6,478	12,444	6,733	9,476	6,213
Ochre, Umber, etc.	10,494	15,532	19,068	17,47[illegible]	13,602	20,103
Oil shale	2,014,860	503 715	2,212,250	608,369	2,361,119	707,177
Petroleum	30	45	35	52	100	150
Phosphate of lime ..	20,000	38 250	18,000	29,500	10,000	20,000
Salt	1,946,496	890,364	2,146,849	1,100,014	2,043,571	976,824
Slates and slabs ...	458,436	1,048,143	434,352	1,027,235	415,029	987,000
Stone, etc.	—	8,670,935	—	8,708,691	—	8,693,743
Sulphate of strontia	5,976	2,988	10,276	5,138	8,061	4,030
Tin ore	13,809	729,213	14,911	782,492	14,488	735,240
Tungstate of soda ...	—	—	—	—	—	—
Uranium ore ...	—	—	22	2,200	31	620
Wolfram	¼	8	104	1,848	138	3,341
Zinc ore	23,202	96,925	22,041	109,890	22,216	113,445
Total values ...		73,476,000		92,794,481		91,238,032

TABLE V.—(*continued.*)

Description of mineral.	1892.*		1893 †		1894.‡	
	Quantity.	Value at the mines and open-works.	Quantity.	Value at the mines and open-works.	Quantity.	Value at the mines and open-works.
	Tons.	£	Tons.	£	Tons.	£
Alum Clay (Bauxit)	7,322	1,860	8,740	4,150	7,970	5,618
Alum shale	2,922	365	2,115	264	3,927	496
Antimony ore ...	6	98	—	—	—	—
Arsenic	5,114	43,686	5,976	57,694	4,801	48,614
Arsenical pyrites ..	4,497	4,988	3,036	2,948	3,288	3,823
Barytes	24,247	29,283	22,343	25,363	20,656	21,410
Bog ore	15,363	7,681	10,747	2,686	7,803	1,951
Clays (excepting ordinary clay) ...	3,103,852	889,375	3,055,208	817,419	3,263,768	823,701
Coal	181,786,871	66,050,451	164,325,795	55,809,808	188,277,525	62,730,179
Copper ore	5 995	11,953	5,346	12,961	5,752½	13,909
Copper precipitate ...	270	3,112	230	2,210	241½	2,313
Fluor-spar	171	188	215	161	126	69
Gold ore	9,990	9,168	4,489	7,6[illegible]7	6,603	13,573
Gypsum	147,540	58,227	143,486	59,369	153,450	66,355
Iron ore	11,312,675	2,970,632	11,203,476	2,827,947	12,367,308	3,190,647
Iron pyrites	13,967	6,957	15,837	7,292	15,523	8,042
Jet	(lbs.) 929	185	(lbs.) 888	177	(lbs.) 479	48
Lead ore	40,024	296,484	40,808	280,539	40,599½	266,995
Lignite	4,247	1,062	3,264	816	334	83
Manganese ore ...	6,078	4 434	1,336	762	1,809	740
Ochre, Umber, etc.	12,131	16,782	10,534	13,880	8,516	14,040
Oil shale	2,089,937	522,484	1,956,520	489,130	1,986,385	496,596
Petroleum	218	409	260	488	49	92
Phosphate of lime ...	12,200	22 250	3,300	5,771	700	1,277
Salt	1,956,524	861,401	1,924,029	735,222	2,235,012	763,629
Slates and slabs ...	418,241	1,025,922	438,993	1,107,626	461,673	1,171,366
Soapstone	—	—	—	—	10	45
Stone	—	8,667,736	—	7 773,743	—	7 695,716
Strontia sulphate ...	5,066	1,266	5,812	2,325	6,823	1,962
Tin ore	14,357	734,565	13,689	637,053	12,910	487,523
Uranium ore ...	37	740	25	500	19	815
Wolfram	125	3,000	22	420	—	—
Zinc ore	23,880	104,016	23,754	81,270	21,821	67,311
Total values ...		82,350,760		70,767,651		77,898,938

* This is the year of the great Coal Strike in the county of Durham.
† This is the year of the great Coal Strike in England south of Durham and in North Wales.
‡ This is the year of the great Coal Strike in Scotland.

The tonnage of coal raised in the United Kingdom in 1895 was 189,652,562.

It may be useful to refer to the mode of occurrence of some of the more important minerals.

Coal has already been mentioned.

Fire-clay is a species of indurated clay. It is found as a stratified bed; it is particularly abundant in the coal measures. As a rule, a bed of fire-clay has a seam of coal overlying it, but the coal may be very thin and of no present commercial value, or may be represented by a carbonaceous shale. Fire-clay is like ordinary surface clay in this respect, that it has no lines of bedding or stratification except where it joins another stratum; nor has it any cleavage, although it has joints along which it can be broken. To be valuable it must be nearly a pure silicate of alumina, mixed sometimes with silica. Fire-clays are seldom worked unless the bed is 2 feet or more in thickness. From 2 to 4 feet is a common thickness for a valuable bed of fire-clay. In some cases there are fire-clays 30 feet in thickness.

Ganister is an unlaminated sandstone. It is nearly pure silica. It often, if not generally, underlies a seam of coal, and is underlaid by fire-clay. It is very hard, and has been often used for road-mending; it is used for making fire-bricks capable of resisting great heat, and also for lining the "holes" in steel-smelting-works. A bed of ganister generally varies in thickness in a short distance from 2 or 3 inches up to 3 or 4 feet.

Gypsum, a white rock used for making plaster of Paris, is found interstratified with the marls of the Keuper formation. The beds vary from a few inches up to about 12 feet in English mines.

Oil shale is found in the Coal Measures of Great Britain, interstratified with other shales and seams of coal. It contains a great deal of petroleum, which is obtained by distillation. It is often got in coal-mining when it happens to be contiguous to a coal-seam; occasionally, as in the case of the celebrated boghead mineral in Scotland, it is worked by itself.

Phosphate of lime, in the shape called coprolites, in England, is excavated in parts of Hertfordshire and Bedfordshire, where it is found, at a little depth below the surface, in some parts of the Cretaceous formation. It is not found in continuous beds, but in hollows or troughs perhaps 7 or 8 feet in depth, which can be easily got with a pick, like a bed of pebbles, and appears to consist of fossilized animal remains such as sharks, etc. In Florida and Carolina it is largely excavated, sometimes in very dry ground, and sometimes dredged from rivers, where it is found in beds. In Carolina the beds are thin, usually from 8 to 18 inches in thickness, though they are occasionally thicker. In Canada phosphates are mined in veins of great thickness.

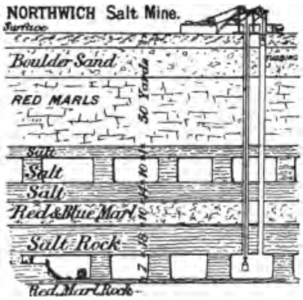


FIG. 35.—Section of salt-mine and strata.

Salt, like gypsum, is a rock, found interstratified with the red shales of the Keuper formation. In the mines of Northwich it is found in beds which attain a thickness of 50 yards (see Fig. 35); in some places, as at Wielitzka, near Cracow, the thickness is upwards of 1000 feet.

Slates are indurated shales. In Great Britain they are generally met with in the Silurian and Cambrian formations. They have been greatly altered by pressure so as to lose the nature and appearance of shale. The valuable slates are those which possess a suitable cleavage for splitting up into roofing-slates. Some of the finer varieties are exceedingly fissile, and an expert workman can split a slab 1 inch in thickness into upwards of thirty full-sized roofing-slates. The seams sometimes attain a great thickness—upwards of 800 feet; and they are often inclined at a very steep angle.

Stone. This is sometimes a stratified rock, as, for instance, Portland, Bath, and other limestones; Whatstandwell (Derbyshire), Bramley Fall (Yorkshire), and other sandstones.

The stratified rocks are sometimes worked by underground mining, as in the neighbourhood of Bath, but as a general rule stone is got by open work.

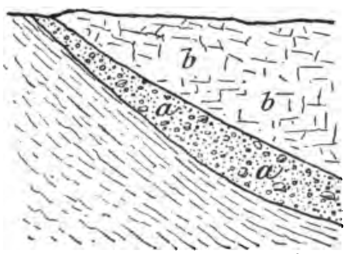


FIG. 36.—Section showing deposit of hæmatite ore. *a*, hæmatite; *b*, carboniferous limestone.

A great deal of stone is unstratified rock of the class commonly called igneous, such as granite, syenite, whinstone. Generally the granite is an intrusive mass, or is in a cone, round which have been deposited more recent rocks. This kind of stone is largely got, both for building and road-making, by open work.

Occurrence of Metals.—Iron is the most important metal got either in Great Britain or elsewhere.

[1] C. C. Hoyer Millar.

It is generally found in the stratified rocks, and to some extent in all, or nearly all, the formations from the most recent to the earliest metamorphic formations.

It is most commonly found as a stone strongly impregnated with iron, and as such is itself a stratified rock. It is also found in great masses of more or less pure oxide of iron filling up cavities in rocks of much older date than the iron ore therein deposited (see Fig. 36).

The chief beds of stratified ore now worked in Great Britain are those of the Oolite and Lias and of the Coal Measures. Till within the last forty years or later, the Coal Measures afforded the chief supply of iron ore after the abandonment of the Sussex workings (Cretaceous rocks).

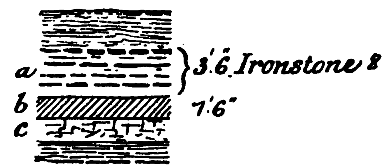


FIG. 37.—Section showing coal-measure ironstone. *a*, shale with black-bed ironstone; *b*, black-bed coal; *c*, seat earth.

Iron ore is found abundantly in most of the British coal-fields, with the exception of those of Durham and Northumberland.

The ironstone of the Coal Measures is found as a bed of hard stone, sometimes reaching a thickness of upwards of 9 feet. Generally it is much thinner, the stone being found in thin beds 1 inch and up to 6 inches in thickness, interstratified with shale (see Fig. 37).

The ironstone is often found as nodules or balls lying close together in regular layers in the shale.

If beds of ironstone of good quality happen to lie in close proximity to a seam of coal, they are often got in the same working, where the cost of working either separately would have been too great.

When thus associated with a seam of coal, the total thickness of ironstone got in one working is sometimes as little as 3 or 4 inches. When the ironstone is got by itself, at least twice that thickness of good iron ore in a total thickness of ore and shale amounting to 3 feet is necessary for profitable working. But these observations as to the profitable working of coal-measure ironstone apply rather to a period of thirty years ago than to the present day. At the present date coal-measure ironstone can only be profitably worked in a few places where it is of exceptional excellence or great thickness, as, for instance, the Blackband iron ore of Lanarkshire; the ironstone of North Staffordshire, worked in a bed 4 feet in thickness; the Black-bed ironstone found near Leeds, which is worked as thin as

4 inches, together with a seam of coal 18 inches thick (see Fig. 37). With these and perhaps a few other exceptions, the coal-measure ironstones, which recently were the chief source of our iron-supplies, are now entirely unworked.

The iron ore of the Lower Oolite has taken the place of the coal-measure ironstone, because it lies near the surface (see Fig. 38), as in Northamptonshire, Leicestershire, Rutlandshire, and Lincolnshire, and is got by open work at about one-fourth of the cost of getting coal-measure ironstone; or else, as in Cleveland, it lies in a bed 9 to 12 feet in thickness, and being also softer than coal-measure ironstone, it can be got by underground mining at about one-third the cost.

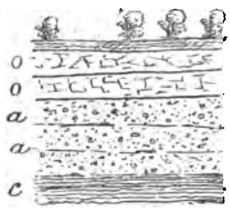


FIG. 38.—Stratified iron ore. *o*, Oolite limestone; *a*, ironstone; *c*, Lias clay.

The other chief sources of iron ore are the masses of red hæmatite found in Cumberland and North Lancashire, and of brown hæmatite in the Forest of Dean and South Wales.

Another oxide (göthite) has been got at a mine near Lostwithiel, in Cornwall. Ore has been found in other places, but out of 2,472,536 tons of hæmatite ore got in Britain in 1891, 2,394,990 tons were got in Cumberland and North Lancashire. The ore in these counties is found in great masses in the Carboniferous (mountain) limestone, measuring in some cases over 1000 feet long, by 750 feet wide and over 400 feet deep, and in other cases in veins 3000 feet long and 90 feet wide. Sometimes the ironstone appears at the surface, but it is often completely buried beneath limestone rocks. It would seem as if the ore had been deposited in caverns formed in the limestone. The larger deposits of ore are sometimes connected by narrow channels filled with ore.

Tin ore comes next to iron in importance amongst the metals raised in Great Britain. Nearly all the British tin-mines are in Cornwall, with slight exception in contiguous parts of Devonshire. Tin is found in veins, in the granite and killas (the latter being a metamorphic clay slate). The veins vary in thickness from a few inches up to 25 feet; in depth they have not been proved, the deepest mine being less than 1000 yards in depth. They are sometimes nearly vertical, and sometimes slope at an angle of 27° from the vertical. Occasionally the veins spread out horizontally into flats. Tin ore is an oxide of tin called cassiterite. Cassiterite is mixed with stone (generally chiefly composed of quartz), forming a rock. At Dolcoath the tin rock is called "blue capel;" it is one of the hardest rocks known. Sometimes, as in the neighbourhood of St. Austell, the vein stone is soft. The ore is sometimes diffused through a large mass of rocks on each side of the vein or leader forming what are called stockworks, as at Carn Brea.

Next to iron, lead is the most widely diffused metal which is mined in this country. It is found in England, Wales, Scotland, Ireland, and the Isle of Man. It occurs in veins, chiefly in the Carboniferous limestone, as in Northumberland, Durham, Yorkshire, Derbyshire, and Flintshire; but there are also lead-veins in the Silurian, as in Mid-Wales, and in other rocks.

The principal ore of lead found in this country is galena, a sulphide of lead. The veins vary in width from a thin leader barely visible to the width of a large cave, say 20 feet. The vein is sometimes full of galena, and often it chiefly contains vein-stuff, that is, spar or other waste material.

The ores of copper, zinc, and barytes also occur in veins, like tin and lead, but they are only produced in small quantities in Great Britain.

English copper was at one time more extensively worked than it is at present, but the price that now rules is hardly sufficient to keep the British mines at work. The production in 1891 was 720 tons of metallic copper, while in 1863 it was 14,247 tons, or nearly twenty times the present output. But in 1863 the average value was £100 a ton, whereas in 1891 it was about £56 a ton. Owing to this low price, most of the British mines have been closed. It may be that at some future date British copper-mines will again be largely wrought. Mines are often closed owing to temporary difficulties which might be overcome if there was a substantial reserve fund with which to carry on the mine during times when the more profitable portions of the veins are exhausted, and it becomes necessary to make further explorations in search of rich deposits. In many cases the profits in good years are paid away to the shareholders, who may be a rapidly changing body, so that when the mine is temporarily less productive, during a period of low prices, there is no fund at hand with which to carry it on. The pumping of the water is often a constant and heavy charge, which cannot be borne by a mine having only a small output. Possibly at some future time, by means of the collective action of the community, the cost of water-pumping will be better distributed, and the total cost reduced in some cases by adits on a low level; improved machinery may be erected for winding, for the conveyance of mineral and material underground and on the surface, for rock drilling, and stamping, a severe system of finance adhered to, and, as a result, the British copper-mines found capable of holding their own in competition with the world.

CHAPTER II.

EXPLORATION.

THE mining engineer may find himself in a country to which he is a stranger, his duty being to ascertain what valuable minerals, if any, exist in the locality, and their mode of occurrence. It would greatly facilitate his inquiries and enhance the value of his discoveries if he had been a diligent student of geological science; and the following pages are intended, not as any substitute for such study, but partly to point out how such study would be useful, as well as the methods he would pursue if his previous study had fitted him for the work he has undertaken. It may

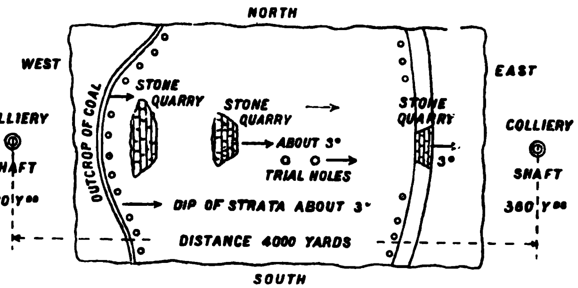


FIG. 39.—Showing plan of estate and vicinity. Bore-holes and trial-holes shown O.

simplify our treatment of this subject if we consider, in turn, some of the numerous problems that present themselves in practice.

Case I.—The particular bit of country he has to explore is in a coal-field. It is indicated in Figs. 39 and 40. The plan and section show the result of the exploration. Two collieries are discovered on the west and east: that to the west being on the "rise" side of the estate, or the side on which the strata crop out; that to the east being on the "dip" side of the estate, or the side on which the strata dip under other and superincumbent strata.

The depths of the shafts are respectively 120 yards and

360 yards from the surface to the lowest seam of coal, which is 4 feet thick. The west colliery is 20 yards above sea-level, and the east colliery 60 yards above sea-level; thus the respective depths below sea-level are 100 yards and 300 yards, as shown in the section figure.

Each colliery is working a similar coal, apparently the same seam, dipping in the same direction at the same rate. A thin seam of coal, 2 feet thick (No. 1 on the section), is found cropping out on the hillside, at a height of 45 yards above sea-level, dipping east at an angle of about 3°; a similar and apparently

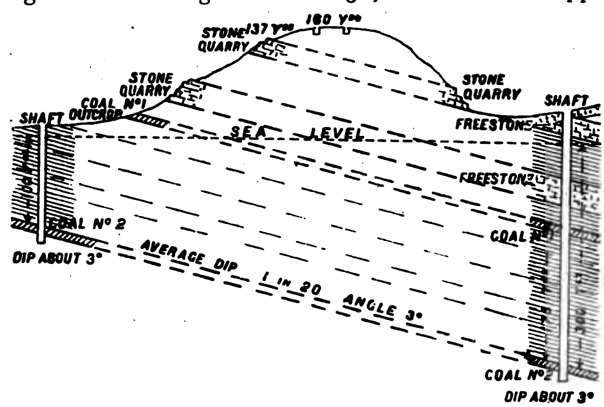


FIG. 40.—Showing section from west to east across the estate. The sketch is "distorted," the vertical scale being about 200 yards to the inch, and the horizontal scale being about 1200 yards to the inch.

identical seam is found in the east colliery shaft, 125 yards below sea-level, and at a distance of about 3400 yards from the outcrop.

It is then noted that a dip of about 3° is approximately equal to a dip of 1 in 20, or 5 per cent.; that is, a dip 1 yard vertical in a distance of 20 yards horizontal. This dip, continued for a distance of 3400 yards horizontal, would give a vertical fall of 170 yards, and that is precisely the fall that is found—45 yards above sea-level added to 125 yards below.

There are also three sandstone quarries in two distinct beds, where freestone is got. The dip of the strata is here ascertained to be about 3° east, and at a corresponding depth the same two beds of rock are found in the east colliery shaft.

On the top of the hill some trial holes are dug 6 or 7 feet deep into the beds of shale, and here the dip is also found to be regular at 3°. The outcrop of coal No. 1 is traced along the hillside across the estate from north to south, either by digging with a pick and shovel, or by boring holes, say 5 yards deep, with boring-rods. The eastern outcrop of the upper bed of rock is also traced across the estate from north to south.

The exploration of the estate is now complete.

The uniformity of the dip from west to east, as proved by the pits, outcrops, quarries, and intermediate trial-holes, shows that there is no fault or break in the strata between the two collieries. The regularity with which the outcrops of coal and rock follow

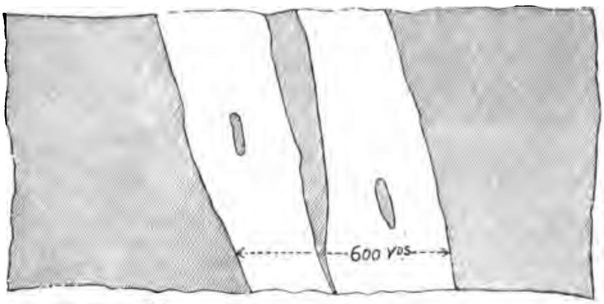


FIG. 41.—Plan showing "wash-out."

the contour of the hill show that there is no fault from north to south. It is, therefore, reasonable to believe that the coal and other strata found in the east colliery continue regularly under the estate, as shown on the section. It is possible that in the distance between the east and west collieries some beds of coal, fire-clay, ironstone, shale, and rock may vary in thickness and quality, or may be in places entirely wanting. It would be very remarkable, indeed, if a coal-seam 4 feet thick were to be entirely

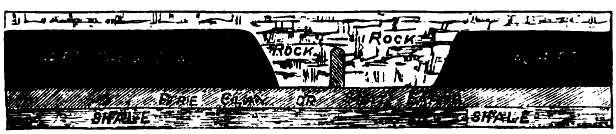


FIG. 42.—Section showing "wash-out."

wanting in such a situation. But it is a contingency to be guarded against. There are "wash-outs," sometimes called "dumb-faults." These are places from which the coal has been washed away as if by some river, and the river-bed afterwards filled up with sand. These wash-outs are sometimes 600 yards in width and 7 or 8 miles in length. Figs. 41 and 42 show a plan and section of a wash-out.

Referring to the plan (Fig. 39), it is possible that such a wash-out may traverse the estate from north to south. If there are no collieries north and south by which the coal is proved, there is no means of ascertaining that such a "wash-out" does or does not exist except by a number of costly borings. Wash-outs are very rare, and, if the estate was in other respects satisfactory,

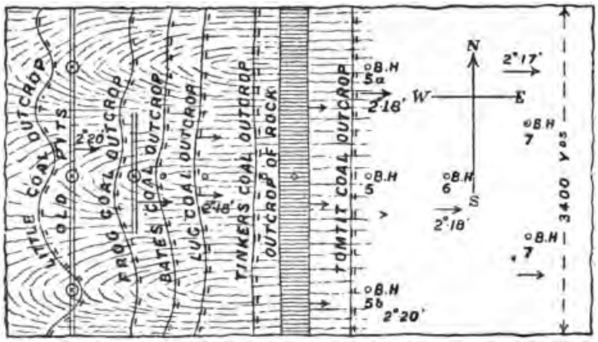


FIG. 43.—Plan of estate, showing outcrops thus =; pits, ⊙; bore-holes, O.

most mining engineers would consider that the coal was sufficiently proved.

Case II. is a coal-field. Collieries are working on the western side only (Figs. 43, 44). The explorer must take all particulars of these depths, inclination of strata, thickness of coal, particulars

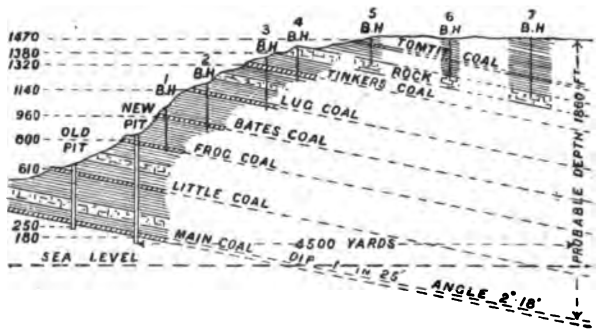


FIG. 44.—Showing section of estate proved by digging and boring.

of the strata found in the shafts, faults, if any, extent of workings, water in the mine or in the rocks above, etc.

Surface-Indications.—Having got all the information that can be obtained at the collieries, the next step is to make a careful examination of the surface, in order to find out what the ground is made of. Sometimes the nature of the ground may be seen in railway cuttings, or in the precipitous sides of a mountain torrent, or in a quarry, or by examining the earth that has been thrown up from any pit or well,—well-sinkers can give information as to coals and rocks found in sinking. A recently ploughed field will often reveal several facts: a broad black band may indicate the outcrop of some dark stratum of shale or of a seam of coal; numerous pebbles may indicate that the strata are

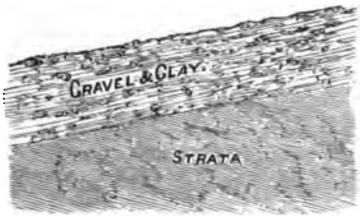


FIG. 45.—Excavation of surface gravel to prove strata.

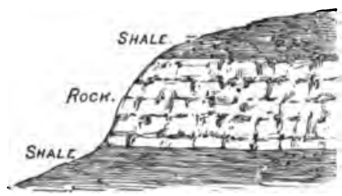


FIG. 46.—Section of hillside, showing shales and rocky cliffs.

covered with a recent deposit of gravel, as in Fig. 45. A steep cliff may be due to a bed of hard rock (Fig. 46), which is able to stand at a steeper angle than shale, bind, or fire-clay, because the weather gradually disintegrates and softens the surface of the shale so that it cannot stand at a steep angle. There are many precipitous cliffs of shale, but the exposed surface of these is constantly falling off. A stream of water issuing from the hillside may form a deposit containing oxide of iron.

A spring of water is often due to a fault diverting an

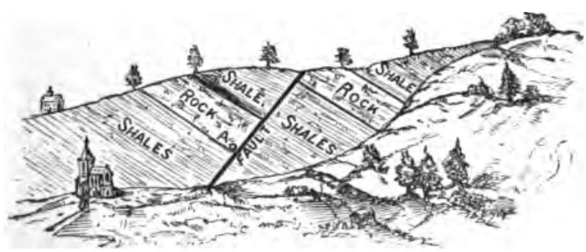


FIG. 47.—Sectional elevation of hillside with fault and spring.

underground current of water. Fig. 47 shows a hillside partly in elevation and partly in section. The rock, having open joints, will conduct the rainfall that reaches the outcrop down to the valley; but a fault crossing the strata throws the rock up, and substitutes a bed of shale, which is impervious to water. The water, being unable to flow downwards to the bottom of the valley, finds its way through the covering of clay and soil on the hillside, and bursts out in a spring at A in the figure. If this spring does not flow off in a clear channel, the ground about it becomes a marsh. For these reasons the presence of a fault may sometimes be inferred from a spring or marsh.

A bright red colour in a ploughed field often indicates the New Red Sandstone formation; and a thick sprinkling of flints generally shows that the chalk rock is below the soil. In the present case the surface gives only negative indications (that is to say, there is no evidence that coal does not continue under the estate), except the outcrop of a bed of rock near the hilltop (see Fig. 44), which seems to be a thick bed of sandstone. There are some old quarries in it where stone has been got for building and for mending the country roads, and the outcrop of this rock seems to cross the estate from north to south, but it is for the greater part of the way covered with soil, and, except where it has been got in quarries, the indications are uncertain.

The workings at the old pits proved that the line of level went due north and south across the estate. The outcrop of the Little coal can therefore be marked on the map by calculation, and the positions so ascertained actually proved by digging holes or by boring. If the ground was flat, the outcrop would go straight from north to south, but as there are hills and valleys, the line of outcrop is twisted—when there is a hill, the outcrop is further west; and when there is a valley, the coal is cut out and the outcrop is further east.

The workings at the new pit have extended a short way north and south, and so far as they have gone they show no change in either the direction or amount of dip. The Frog coal crops out near the pit, as proved by actual digging; the position of the outcrop across the estate is then marked on the plan by calculation, and then the exact places ascertained by digging and boring.

In this case the line of outcrop is parallel to that of the Little coal. On the hillside above the new colliery a bore-hole is next started, and at a depth of about 10 yards a new seam of coal, named Bates's coal is met with (named after the foreman who is in charge of the boring); the bore-hole is continued down to a depth of nearly 60 yards, when it has passed through the Frog coal into the strata below.

The outcrop of Bates's coal is now traced across the estate. Another boring (No. 2) is made on the hillside above No. 1, at a depth of five yards; it proves another new seam of coal, which is named Jug coal (because the workmen's tea arrived just as the coal was found). This boring is also continued down until some previously known strata are found, and within a depth of 65 yards Bates's coal is bored through.

The outcrop of the Jug coal is traced, and boring No. 3 is made; and subsequently borings Nos. 4, 5, 6, 7, and 8, as shown in Figs. 43, 44.

In this way a succession of new strata, 1260 feet thick above the Frog coal, is proved. It is also shown that the dip is regular towards the east, and that there are no faults to disturb the line of outcrop.

There is now every reason to believe that the main coal underlies the whole estate, and will be found at a depth of 620 yards below the surface at its eastern boundary, with six other seams of coal above it in regular succession. The total amount of boring done in proving this estate is 800 yards, exclusive of a number of shallow borings or excavations along the line of outcrop.

The estate might have been proved in three other ways.

(*a*) By driving downhill in the main coal from the new pit; this would take about six years at the ordinary rate of driving.

(*b*) By boring one deep hole where No. 7 hole is; but the boring of deep holes involves a considerable outlay, and there is great risk that the results of a solitary boring may mislead.

(*c*) By sinking a pit; but this is a costly operation, and the exploration above described is a preliminary operation to discover if there is sufficient justification for such a large outlay.

If the exploration has to be made in a short time, this may be accomplished by employing a large number of men, and boring at five or six places at the same time.

Case III.—Here is a case where the hasty observer may be easily misled (Figs. 48, 49). The east and west collieries are

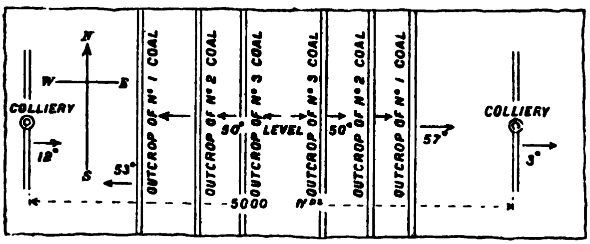


FIG. 48.—Plan of a coal-field.

each working the same seam, which dips the same way, and it is quite reasonable to suppose that the coal No. 2 may continue all the way along the line A A in the figure. The colliery shafts and the workings show nothing to suggest any other conclusion.

But the careful inquirer examines the whole country before coming to a decision, and in this case he discovers that the strata are very highly inclined at some points between the two collieries, and that the strata dip in some places towards the west, whilst the general dip is east. It is necessary to continue digging and boring till the outcrops of the seams of coal have been proved

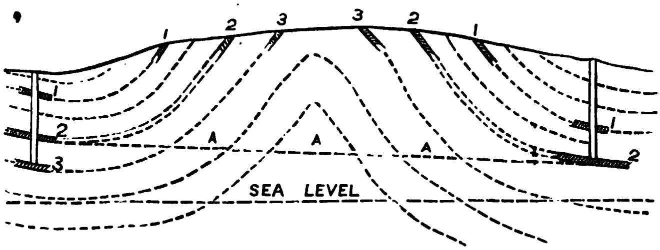


FIG. 49.—Section showing anticlinal ridge.

and traced. In this case the No. 2 coal is the lowest workable coal-seam, and a great part of the estate is therefore shown to be barren.

Case IV.—There is only one colliery, on the west, and all the country on the east of this colliery is unknown. By examining the surface and by digging holes and making shallow bore-holes, it is discovered that there is a change of dip between A and B (Figs. 50, 51). A bore-hole at A gets into some ground which is consistent with being in a fault, and the bore-hole at B, being put down to a depth of 100 yards, passes through a seam of coal

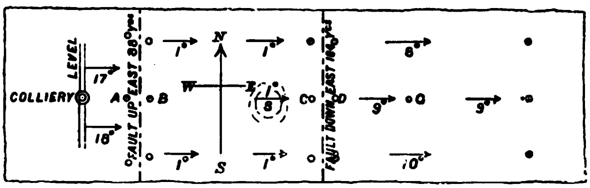


FIG. 50.—Plan of estate: faults shown — - — - —; bore-holes, ° ° °.

which is recognized as No. 2 coal. The mode of recognition can be better understood by reference to Fig. 52.

This is a section of the strata as proved by the bore-hole at B. This section corresponds almost exactly with the section found at the colliery shaft, where No. 2 coal has the same very hard black shale roof with ordinary shales above; it has the same fire-clay floor and shale below, and the same rock 48 feet thick below that, and the coal-smut near the surface of the bore-hole corresponds to No. 1 in the shaft. At the hilltop at the place S on the figure there is also found a coal-smut, or traces of a seam of coal, just where No. 1 coal would be if the coal in the bore-holes B and C were No. 2 coal.

Another bore-hole at C gives a similar section to B. A bore-hole at D gives a very different section, being in a bed of

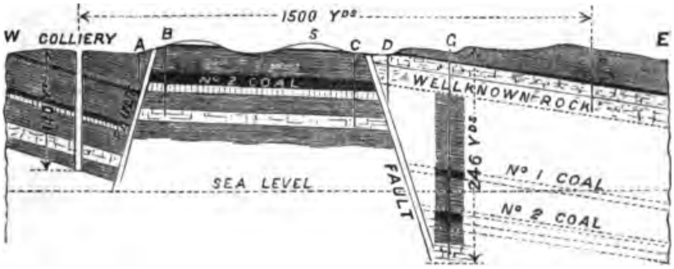


FIG. 51.—Section of estate, distorted (not drawn to scale).

sandstone. It is suspected that this is the "well-known" rock which is found in a quarry 2000 yards due north of the colliery, and which has there been proved to be about 80 to 90 yards above No. 1 coal. Acting on this theory, a deep bore-hole is put down at G, which proves coal Nos. 1 and 2 as shown in the figure. Other bore-holes of little depth as shown on the plan prove the regularity of the dip and the lines of the faults originally discovered by the holes at A, B, C, and D. The No. 2 coal was expected to be 162 yards deep at B, but it is actually only 43 yards deep. This alteration in depth could not be altogether due to a change in the inclination of the strata, the dip being proved by observations in shallow holes, and the change in dip, as subsequently proved at B and C, would reduce the anticipated depth of No. 2 coal to 131 yards. The difference between 131 yards and 43 yards can only be accounted for by a fault, and the fault is an upthrow to the east of 88 yards. Fig. 53, drawn to a natural scale, shows these faults.

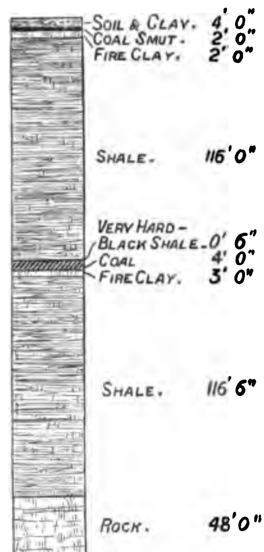


FIG. 52.—Section of bore-hole. Scale 100 feet to 1 inch.

As a rule, a geological horizontal section is either sketched to no scale at all or is drawn to a "distorted" scale. The vertical scale is different to the horizontal scale, because the depth of a section is generally so much less than its length, that, if both are drawn to the same scale, either the details of the vertical section are invisible or the length of drawing becomes unwieldy. Thus a section might have a vertical scale of 40 feet to 1 inch and a horizontal scale of 400 feet to 1 inch. But upon such a section the angle of dip cannot be correctly shown. Therefore when it is necessary to draw the strata dipping at their real or natural angles, the section must be drawn to a natural scale, that is to say, the horizontal and vertical lines must be drawn to the same scale.

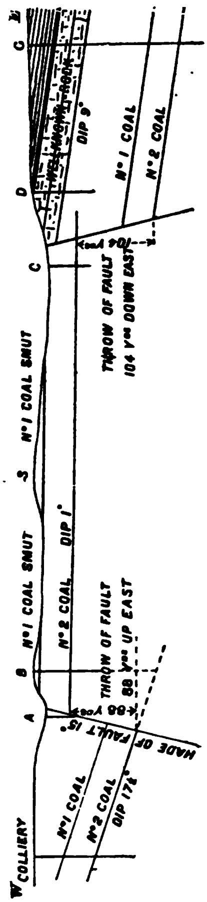


Fig. 53.—Section on the same line as Fig. 51, but drawn to a "natural" scale of 266 yards to 1 inch.

Case V.—In this case there are no pits. By careful search, digging, and boring, two seams of coal are discovered. No. 1 is 3 feet thick, No. 2 is 4 feet, and has 6 inches of cannel on the top, by which it can be easily distinguished from No. 1 coal. No deep borings are made; the deepest bore-hole is 10 yards. Forty or fifty excavations or borings showed the outcrops, as in Fig. 54, and the inclination of the strata to be uniform, as shown in the section (Fig. 55), about 26½° from the horizontal, or 1 in 2. The surface of the ground being quite level and the inclination uniform, the line of outcrop should be quite straight. But in this case the line of outcrop is broken, as shown in the plan (Fig. 54), and this variation in the position of the outcrop can only be accounted for by faults. The line of outcrop A is thrown back to B, a distance of 86 yards on the plan, and as the inclination is uniformly 1 in 2, it shows that there is an upthrow fault to the south of 43 yards as shown on the cross-section $x\ y$ (Fig. 55). At C the line of outcrop is thrown forward 36 yards, proving a down-throw fault to the south of 18 yards. At D the outcrop is thrown back again 70 yards to the east, proving an upthrow to the south of 35 yards.

Case VI.—In this case the ground is not level, but, except for a slight irregularity, the outcrop keeps on a line nearly north and south until getting near the point x (Fig. 57), when it turns to the east. At x the line of outcrop again turns towards the south; the dip is ascertained to be uniformly 26½°, or 1 in 2, east. The hill at C is 25 yards above the ground at A, and this extra cover ought to have carried the outcrop further west 50 yards, as

Fig. 54.

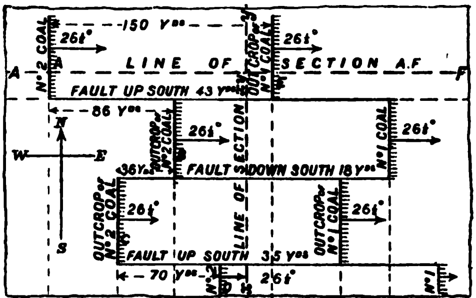


Fig. 55.

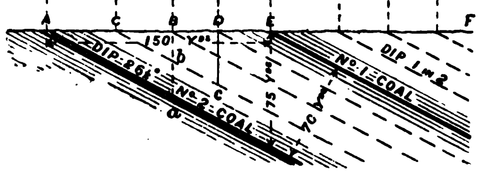


Fig. 56.

FIG. 54.—Plan of estate, showing outcrops of coal and dip of strata. FIG. 55.—Line of section x y, showing three faults. FIG. 56.—Section on line A F, showing effects of faults by the dotted lines. Drawn to a natural scale of 140 yards to 1 inch.

shown in the section (Fig. 59) at C′. But as the outcrop is not thrown forward (and the dip is unchanged), it follows that there is an upthrow fault to the south 25 yards, as shown in the cross-section (Fig. 59). When the low ground is reached again at x, the outcrop is found to have gone back 50 yards eastwards from the point A: this is due to the fault.

Case VII.—In this case (see Figs. 60 and 61) the coal-field appeared to terminate on the western side of the estate, the centre and the eastern side being overlaid by Permian rocks and New Red Sandstone.

No. 2 coal was the lowest coal-seam of any value, and, so far as could be seen, only the lower and unproductive coal measures were likely to lie to the eastward. Bore-hole No. 1 was, however, put down, and passed through two coals similar to Nos. 1 and 2. Bore-hole No. 2 was then put down, and passed through the same seams of coal, showing that between these bore-holes and the colliery there is an anticlinal ridge, and that valuable coals underlie the eastern side of the estate. In this case the surface rocks, being those of an unconformable formation, give no indication whatever of what may be below, except in this one respect, that the surface rocks being of a newer formation than the coal measures, it is, of course, always possible that the coal measures may be below.

Fig. 59. Fig. 57.

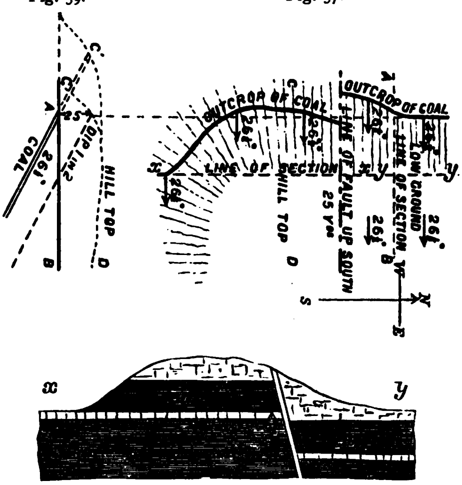


Fig. 58.

FIG. 57.—Plan showing outcrop of coal. FIG. 58.—Section on line x y, showing hill and valley, seam of coal, and fault. FIG. 59.—Section on line A B, showing effect of hill and fault on line of outcrop.

The previous seven cases refer entirely to stratified minerals, whether coal, ironstone, fire-clay, building stone, gypsum, or other mineral.

The search for minerals such as tinstone, lead, copper, gold, silver, etc., is generally conducted in a different manner. It is seldom that the search for a valuable mineral is undertaken without some guide as to where it is likely to be found. In Europe, Asia, and Africa, modern workings of these minerals are generally a continuation of ancient workings, the discoveries

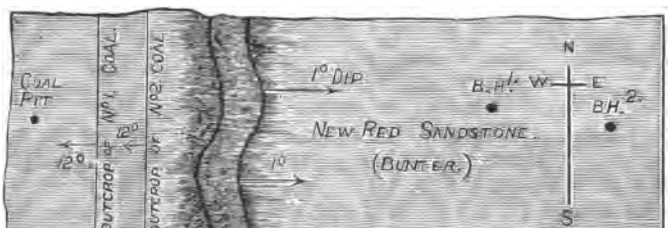


FIG. 60.—Plan showing estate partly covered by the Permian and New Red Sandstone formations.

gradually extending. In Australasia and some parts of America the mines seem to be entirely recent discoveries. The original discovery of valuable ores in any district is often, if not generally, accidental rather than the result of elaborate scientific investigations.

These minerals are generally found in hilly or mountainous

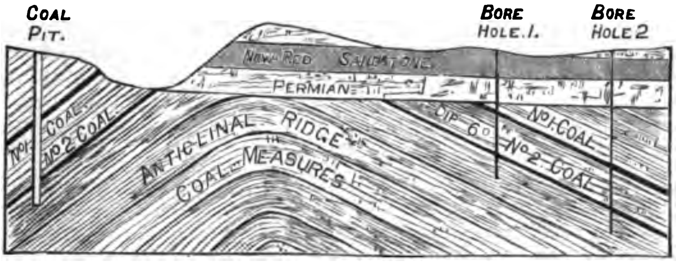


FIG. 61.—Section showing an anticlinal ridge, and eastward extension of coal-field entirely concealed by new formations, and only discoverable by boring.

districts (in Germany the same word *berg* signifies "mountain" and "mine," and *bergman* means "miner"). There are probably several reasons for this. One reason is at once apparent—it is that excavations made on the side of a hill can be drained without pumping machinery. Before the days of steam-power this would be sufficient of itself to limit mining operations to the hills as a general rule, and even at the present day the preliminary operation of searching for minerals would be rendered too costly if pumping plant had to be erected before it was known that there was any large quantity of mineral to get. Another reason is also apparent, and that is that a mineral vein is more likely to be seen on a hillside off which the soil is washed by wind, rain, and torrent, than on a plain.

Exploration for Gold, Tin, etc.—Some of the minerals, such as gold, tin, lead, are found in recent deposits of gravel, such as old river-beds, having evidently been washed down from the higher ground. The usual mode of exploration is to dig up this gravel, and put some in a "pan" with water; then to pick out all the clean pebbles, leaving only the sand and mud; the muddy water is carefully poured off, leaving only sand; the lighter particles of sand are stirred up in the water and poured off; the residue at the bottom of the pan is then carefully examined to see if it contains gold. There are many ways of working the pan, some depending on the supply of water. If gold is found, the gravel may then be washed more systematically and cheaply.

River-Deposits lead up to the Veins in the Hills.—If a mineral is found in river gravel, it seems almost certain that there must be a store of this mineral in the hills from which the river flows, so that the explorer naturally follows the stream up to its source. It does not follow that it will be profitable to work the mineral as it lies in the solid rock. It may be that the auriferous or stanniferous deposit in the river-bed is the more valuable mine, because here the rocks have been broken up by natural agencies, thus leaving to the miner only the work of sifting. It may be also that there has been in some cases a natural process of concentration (akin to the panning process), some of the detritus having a great deal more than its due proportion of metal, and other parts having little or none; thus it is possible that the digger, lighting on a concentrated deposit of ore in the ground, may get rich, but, on attacking the ancient rocks from which the metal has come, may find his labour altogether unprofitable.

Case VIII.—Discovery of Veins.—The miner in search of a mineral vein looks about for joints or cracks in the rocks, or for projecting knobs or reefs, because the vein, being composed of different materials to the rocks through which it passes, will probably be either softer or harder, more easily broken up or more enduring than the rocks on either side, which are technically called the "country rocks;" thus it follows that the existence of a vein is often marked by a depression or fissure, or by a ridge or reef. Very often the vein is distinguished by colour from the country rocks; a quartz vein (containing gold) is white, so also is a calc-spar vein (containing lead and silver).

Having discovered a mineral vein, some portion must be broken off by wedging or blasting. On a clean surface of veinstone the mineral may often be seen; lead ore may appear in great masses, and so may copper ore and other minerals; gold is often visible in specks, sometimes very minute; if the gold is invisible, the veinstone will have to be crushed and assayed. The same observation applies to tin, which is often minutely divided in a very hard rock.

Case IX.—In this case veins have been already discovered and worked in the district between A and B (Fig. 62). The veins

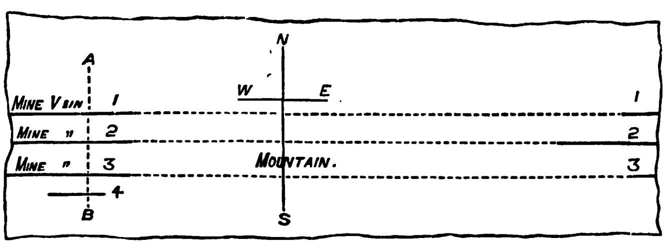


Fig. 62.—Plan showing three veins.

run from west to east; a search is therefore made to see if the veins continue in the same direction. At the eastern side of the mountain three very similar veins are found; it is thought that they are very likely the same veins. Fig. 64 is a section on the line A B, showing the veins.

Case X.—Here there are four veins worked at the western side of the estate. Two of the veins run east and west, and two from south-west to north-east. Numerous excavations made show that they probably continue as shown by the dotted lines on Fig.

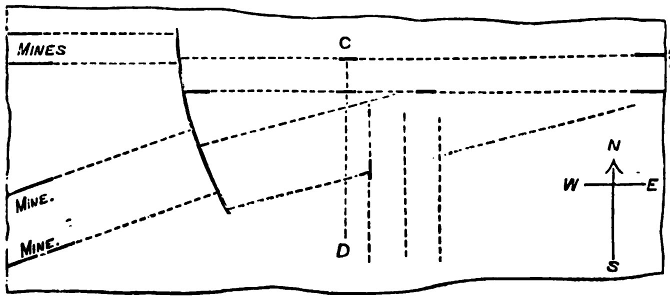


FIG. 63.—Plan showing veins crossing and branching.

63. Subsequent workings in these veins show that they are inclined at angles of from 60° to 75° from the horizontal, and pass downwards through the killas into the granite, as shown in Fig. 65.

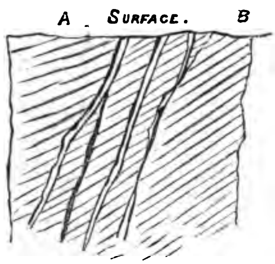


FIG. 64.—Section on line A B, Fig. 62, showing mineral veins.

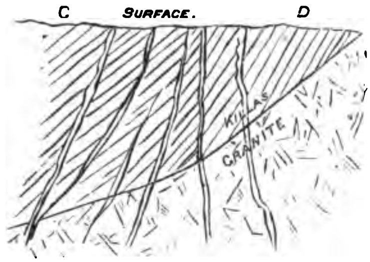


FIG. 65.—Section on line C D, Fig. 63, showing veins crossing two formations.

Case XI.—In this iron ore was accidentally discovered cropping out at A, and, being followed, a large deposit was found. Subsequently, borings were made as shown on the plan and section (Figs. 66, 67). Some of these passed through the ore, and some did not; in this case the only rule to follow was to bore in the limestone. In some instances the limestone is overlaid with shale; in that case the bore-hole will have to be so much deeper to pass through the limestone.

In districts where the ore occurs in similar water-made caverns, the rule is to make the explorations in those strata which are likely to have such caverns or large fissures. In some cases the deposits of ore are found to follow some well-marked line of fault or fracture; in other cases they follow some line of

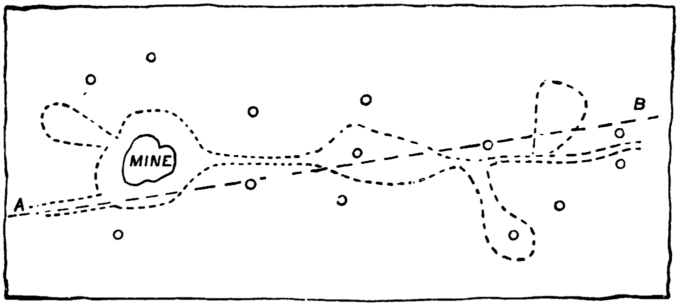


FIG. 66.—Plan showing iron-mine, and borings shown °; ultimate extent of iron ore pockets shown by dotted lines.

upheaval, where there is no actual fracture or slip; in still other cases it is difficult to lay down any rule for guiding the explorer.

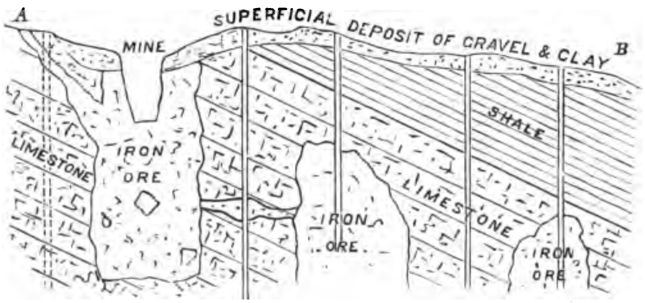


FIG. 67.—Section on line A B, showing iron ore deposits in limestone rock and bore-holes.

Details of Methods of Exploration.—A good deal has been said in preceding pages about ascertaining the dip or inclination of the strata, and the proving of outcrops by digging and boring. The following paragraphs show in a little more detail how this may be done.

Fig. 68 shows an excavation 9 feet deep, which exposes the outcrop of a seam of coal. On the surface of the coal is laid a straight-edge, and on that is put a clinometer, and the angle of the dip is read.

Fig. 69 shows a clinometer. The arc is graduated in degrees. It may also be marked to show the dip in percentages (see Fig. 70).

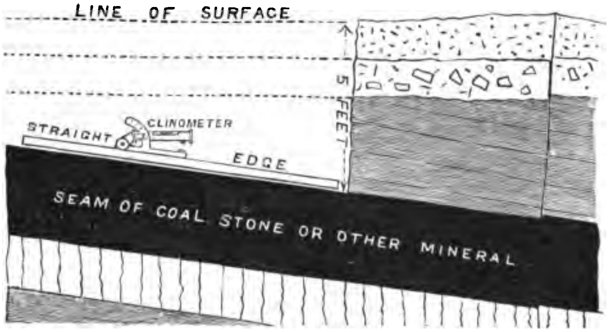


FIG. 68.—Showing excavation to prove outcrop and dip (about 7½°, or about 1 in 7½).

1 in 100 is 1 per cent.; 1 in 50 is 2 per cent.; 1 in 10 is 10 per cent., etc.; 1 in 1 is 100 per cent. Thus 45° would be marked 100 (per cent.); 7½°, 13 (per cent.); 5°, 8·7 (per cent.), etc.

Fig. 71 shows how the dip may be obtained by means of a common mason's level (either with a plumb-bob or a spirit-level), a straight-edge, and foot rule.

Fig. 72 shows how the dip may be ascertained by means of an excavation and bore-hole, and taking into account the difference in the altitude of the surface.

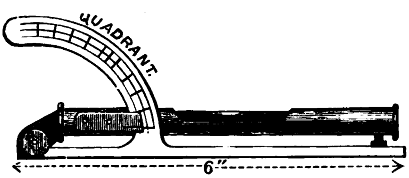


FIG. 69.—Clinometer, scale one-third full size.

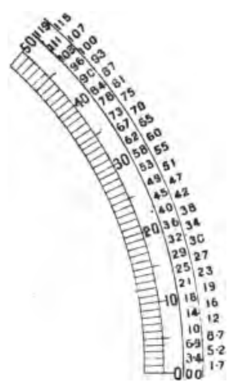


FIG. 70.—Part of quadrant, full size. The inner row of figures are degrees; the middle row are percentages of inclination for even numbers; the outer row for odd numbers.

It is not always possible to ascertain the true dip by one observation; it often happens that it must be ascertained from two observations, neither of which is on the line of greatest dip. Fig. 72*a* shows in plan two lines along which the dip has been observed: C D, direction south-east 50°, dip 1 in 10; E F, direction north-east 30°, dip 1 in 20. These two lines must be plotted on paper to scale, showing their direction and position correctly. The lines F E and D C must then be prolonged till they meet in G. On the line G D must then be marked out a length of 10, G H (because the dip is 1 in 10), and on the line G F a length of 20, G I (because the dip is 1 in 20); G H and G I must be connected by the line H I, and a perpendicular to this line drawn

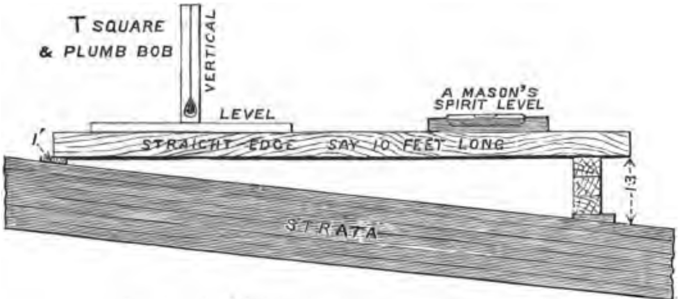


FIG. 71.—Showing plumb-bob level, spirit-level, straight-edge, strata, and dip (1 in 10, or $5\frac{3}{4}°$).

from the apex G to K in H I; G K is then the direction of the greatest dip, and represents the amount of dip. The length G K is $8\frac{1}{4}$, and therefore the inclination is 1 in $8\frac{1}{4}$.

In a similar manner the true dip may be ascertained from the depth of three pits, represented in Fig. 72*a* by the letters G, D, and F. It is first necessary to reduce the actual depths to their

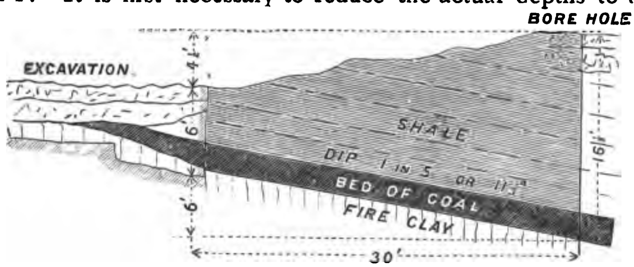


FIG. 72.—Shows outcrop and inclination proved by excavation and bore-hole.

relative depths above or below the sea-level. Thus if G is 150, D 220, and F 250 yards deep all down to the same coal; if the top of G pit is 300 feet above the sea-level, D 360 feet, and F 390 feet, then 60 feet must be taken off D, and 90 off F, reducing the depth of D to 200 yards, and of F to 220 yards. Then if the distance between G D and G F is known, the rate of inclination on those lines can be calculated, and the true dip set out in the manner given in the first instance.

If the dip is ascertained by means of a clinometer and is recorded in degrees, the rate of inclination can be quickly ascertained by bearing in mind that it is equal to the ratio of the radius of the circle to the cotangent of the angle of inclination. If, for instance, the radius is 1 and the cotangent of the angle 10, the inclination would be 1 in 10; for an angle of 6° the (natural) cotangent is 9·5, and the inclination is therefore 1 in 9·5.

For the geological exploration of a country it is necessary to have a map and plan showing the position of all trial holes, quarries, outcrops, hills, rivers, etc. And it is also necessary to have the relative altitudes of all the places. The explorer must

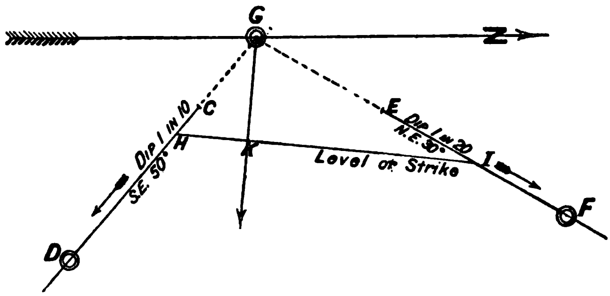


FIG. 72*a*.—Method of finding true dip.

therefore either be himself a surveyor or he must have the assistance of a surveyor.

In exploring for the first time a large new country, a party is generally substituted for a single individual, and this party includes competent surveyors. In civilized countries the mining engineer can generally obtain maps and plans on which he can mark with accuracy the mines, excavations, bore-holes, etc., by means of a few measurements from the nearest landmarks, such as fences, roads, rivers, buildings, etc. He must then, by the aid of a surveyor's spirit-level, take the relative altitudes of the mines, excavations, etc. It will facilitate his explorations if the contour lines are marked on the map—that is to say, lines following all the sinuosities of the hillsides, which throughout their length mark places of equal altitude above sea-level; between each line the ground falls or rises say 25 feet. These contour-lines are marked on some of the ordnance maps, and are of great service to the geologist.

As some of the readers of this treatise may not be acquainted with the use of angles, the following paragraphs will be useful to them.

The circumference of a circle is divided into 360 equal parts, called *degrees*, written thus, 360°; each degree is divided into 60 equal parts, called *minutes*, written thus, 60′; each minute is divided into 60 equal parts, called *seconds*, written thus, 60″. Any line drawn from the centre of a circle to its circumference is called a *radius*. Any line drawn across the circle from side to side through the centre is called a *diameter*, and is, of course, equal to twice the radius. Any two lines drawn from the same point contain an *angle*. Any two lines drawn from the centre of a circle to the circumference contain an angle, and the size of the angle is measured by the number of degrees between the two lines on the circumference. Thus in Fig. 73 the two lines A B and A D contain an angle, C, which is measured on the circumference, and is there seen to be about 26½°, or more exactly 26° 34′. If from the point D (in the line A D) a line, D E, is drawn vertically to and touching

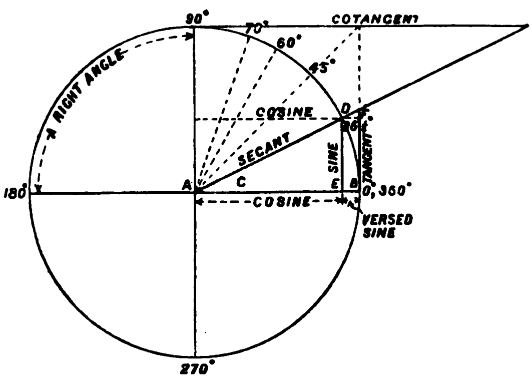


FIG. 73.—Circle divided into parts.

the line A B, the line D E is called the *sine* of the angle C. That part of the line A B which is between E and A is called the *cosine* of the angle C. In the same way, sines and cosines can be drawn for any other angle not exceeding 90°. When an angle has 90° it is called a *right angle;* the two sides of a right angle are perpendicular to, or square to, each other. That part of the line A B which is between E and B is called the *versed sine* of the angle C. That part of the circumference which is between B and D is called the *arc*. If from B a line is drawn perpendicular to A B until it meets the line A D produced to F, the line B F is called the *tangent* of the angle C, and the line A F the *secant* of the angle.

If any number of degrees are taken from a quadrant, the number remaining to make up 90 are called the complement. This complement has also a sine, tangent, and secant, which are called the cosine, cotangent, and cosecant. That part of the radius which is equal in length to the sine of the complement is commonly called the cosine, as shown above (see Fig. 73).

If the line A D is taken to represent the inclination of a seam of coal, and the line A B the level line, then the arc measures the angle of inclination, and it is found to be 26° 34′. Then the line D E, which is the *sine*, represents the vertical fall or rise of the slope A D; and the line A E, which is the *cosine*, represents the horizontal length between the points A and D. Therefore the vertical rise or fall : horizontal length : : sine : cosine; or $\frac{\text{horizontal length}}{\text{vertical (rise or fall)}} = \frac{\text{cosine}}{\text{sine}}$.

The value of this rule (the truth of which is self-evident) is to be found in the fact that the relative lengths of the sines and cosines of every angle from 0° to 90° have been carefully calculated, and are given in every book of mathematical tables.

If the reader has such a book, and will look under the head of "Natural Sines, Cosines, etc.," he will find a page headed "26°," and under this figure he will find, rather more than halfway down the column, 34′; opposite this latter figure, and in the column headed "Sine," he will find the figures 4472388; under the column headed "Cosine" he will find the figures 8944146. These figures represent the relative lengths of the sine and cosine; or $\frac{\text{cosine}}{\text{sine}} = \frac{8944146}{4472388}$; or, striking out the last three figures from each, $= \frac{8944}{4472} = \frac{2}{1}$; that is to say, the horizontal length is twice as much as the vertical fall (or rise); or the slope is 1 in 2, or 50 per cent.

Instead of using the table of sines and cosines, we may calculate in another way. Assume that the line A F, or secant, represents the inclined surface; then the line F B, or tangent, represents the vertical rise or fall, and the line A B represents the horizontal length between the points A and F. The line A B is the radius. Referring now to a book of mathematical tables, under the head of "Natural Tangents and Secants," we shall not find the proportional length of the radius given, because it is always taken as one; but under the head of "Tangents," and under the column "26°," and opposite the figure 34 in column of minutes, will be found the figure 0·5000352; therefore $\frac{\text{radius}}{\text{tangent}} = \frac{1}{0{\cdot}5000352}$; or, leaving out the last three figures, $= \frac{1}{0{\cdot}5000} = \frac{1}{0{\cdot}5} = \frac{1}{\frac{1}{2}} = \frac{2}{1}$; or the horizontal distance which is radius is twice the vertical distance which is tangent; or the slope is 1 in 2, or 50 per cent. (see p. 49 for use of cotangent).

In case there are no mathematical tables at hand, the angle may be marked out by a protractor. If there is no protractor at hand, a circle may be drawn with a bow-pen (or pair of compasses), and divided into quadrants; each quadrant may be bisected, and then divided in 45 equal parts, each of which will be a degree. Another and an easier way is to draw a circle or part of a circle, and with the compasses open at the same length as radius, mark off chords on the circumference (see Fig. 74). A chord equal in length to the radius covers an arc of 60°; so that six such chords complete the circumference. Each arc, therefore, covers 60°; if it is bisected, each division contains 30°; this can easily be divided into three nearly equal parts each of 10°, and these again into single degrees.

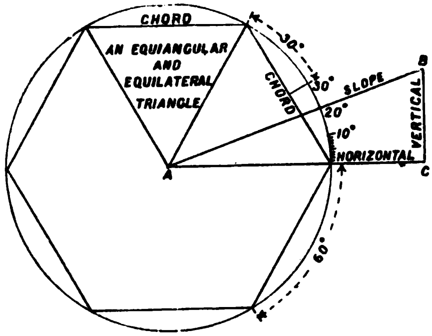


FIG. 74.—Circle divided by chords.

The slope being set out at the ascertained angle, 20°, as A B (Fig. 74), for the length of the incline, say 250 yards, represented by the length A B, then B C can be drawn vertical to A C the horizontal line, and B C and A C may then be measured with a scale, and are approximately 85 yards and 235 yards, or to find the percentage; or, as $235 : 85 :: 100 : x$; $x = \frac{8500}{235} = 36$; or the slope is 36 per cent. On looking at the table of tangents, it will be seen that the exact percentage is 36·39702.

The equipment of the explorer should contain a good hammer, a magnifying-glass, a small bottle of acid to test for limestone, a clinometer, an aneroid barometer and thermometer to measure altitudes, a compass, a note-book, and a map. For detailed explorations, the services of stout labourers with shovels, pick-axes, drills, blasting powder (dynamite, etc.), boring-tools, etc., will be required.

CHAPTER III.

BORING: HAND, STEAM, RODS, ROPES, TUBES, FREE-FALL, DIAMOND, ETC.

BORE-HOLES are in many cases the best means of completing the exploration of mineral estates.

A pit shows the ground more plainly, but in the majority of situations a pit cannot be sunk many feet deep before water finds its way in, and then the cost of sinking becomes too great for the purposes of exploration. Apart altogether from water, the cost of making a small hole from 3 inches up to 12 inches in diameter is generally very much less than the cost of sinking a large pit from 6 feet to 16 feet in diameter. The bore-hole may also be put down more quickly.

Two Kinds of Boring.—There are two chief modes of boring. One is by a percussive drill, which chips the rock into small fragments, subsequently removed; and the other, by a rapidly revolving ring, which grinds the rocks into powder. Of the percussive method of boring there are many varieties. Of the grinding method there are only two varieties; one of these is the diamond drill, and the other is a similar tool, but using hardened steel instead of diamonds.

Percussive Boring.—Various Methods.—The most common method of boring in England for depths of from 5 to 50 yards (a method used in some cases for depths of several hundred yards) is by means of a steel chisel screwed on to iron rods, suspended by a spring-pole. The chisel is steel welded on to the end of an iron rod, making a total length of 18 inches. It is a single blade, for hard rock a cross-blade, as wide as the intended diameter of the hole, say from 3 inches up to 8 inches. Unless it is intended to go very deep, the holes are seldom started more than 6 inches in diameter.

The rod varies from $\frac{3}{4}$ inch square up to $1\frac{1}{8}$ inch square; $\frac{7}{8}$ inch and 1 inch are common sizes. The rods must be made of the very best quality of iron that can be obtained, so as to diminish the chances of fracture. They are screwed together the socket end downwards. There are two sets of shoulders at the upper end of each rod, on opposite sides (see Fig. 75, 1, 2, 3, 23).

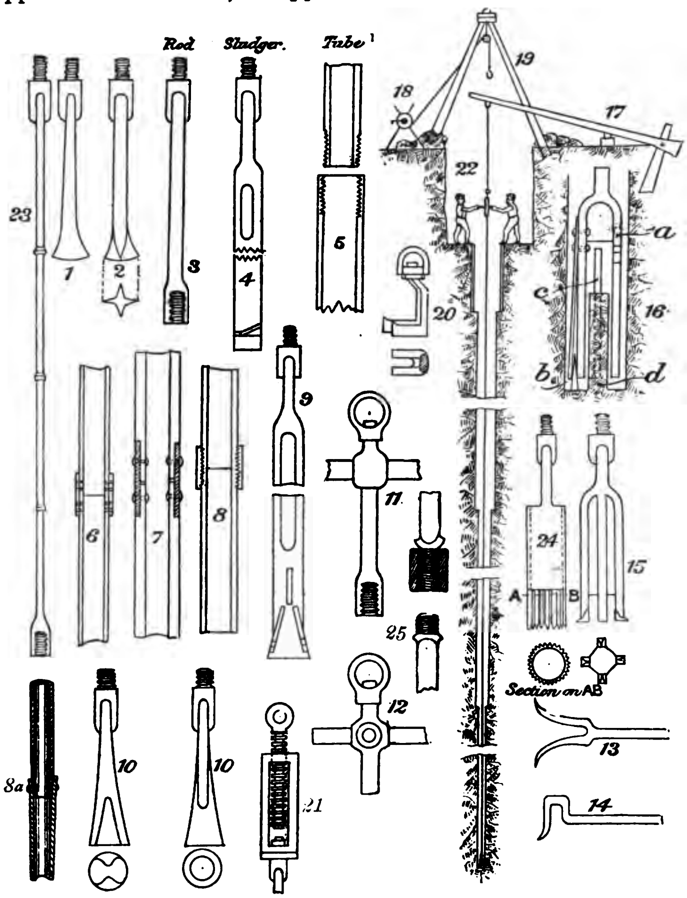


FIG. 75.—Hand-boring tools. 1, single chisel; 2, cross chisel; 3, rod; 4, sludger (6 ft.); 5, screw-jointed tube; 6, tube with outside collar; 7, tube with flush joint riveted; 8, tube with screw socket; 8*a*, riveted socket joint; 9, grappler (6 ft.); 10, reamers; 11, single cross-head; 12, double cross-head; 13, fork; 14, key; 15, 16, core cutter and extractor; 17, spring-pole; 18, windlass; 19, legs and pulleys; 20, hook to lift rods; 21, temper screw; 22, excavation and hole; 23, rods; 24, core-cutter; 25, rods jointed with loose socket.

By means of these shoulders the rod can be lifted or suspended.

[1] These flush-jointed tubes are the same diameter inside and outside through the whole length of tubing.

The set of boring-rods consists of one chisel ½ yard long, rod ½ yard, rod 1 yard, rod 2 yards; the remaining rods are of lengths which under the particular circumstances are most convenient, say 2 yards, 3 yards, 4 yards, 6 yards, or even more. The rod must not be longer than can be conveniently transported, handled, and suspended from the frame with the lower end above the bore-hole. Four yards is often a convenient length; they are, however, often made in 2-yard lengths; these can be screwed up into 4-yard or 6-yard lengths as required. A loose socket-joint is shown in Fig. 75 (25).

The process of boring is as follows: If the hole is probably to be a shallow one—from 3 to 15 yards—the bore-hole is begun with the chisel screwed on to the shorter rods without any preliminary work, simply putting on the ground a piece of plank with a hole in it, through which the chisel is passed; the boring is made by two or three men. Rods are added as the hole gets deeper, up to a depth of 15 yards; below that depth four or five men are required, and also a windlass and pulley frame, as shown in Fig. 75 (18, 19, 22). After 30 yards a spring-pole or other contrivance is necessary to sustain the weight of the rods. Fig. 75*a* shows also a method of boring by lever instead of spring-pole, and also a windlass and frame. These frames are now often made of iron tubes or bars. If the hole is probably to be from 20 to 50 yards, it is started more carefully. It is a common practice to dig a pit 6 or 7 feet in diameter and about 7 feet deep. The digging of this pit serves several purposes. In the first place, it can be done as quickly and cheaply as an equal length of boring at a depth of over 25 yards. In the second place, it clears away the loose ground at the surface, which might interfere with the boring. In the third place, it has the effect of additional height on the pulley-frame, just making a short pulley-frame available for drawing the rods in lengths of 4 to 6 yards. In the fourth place, the pit shelters the men from storms.

At the bottom of the pit a smaller hole is hacked up with pick-axe and drill, into which an iron pipe can be placed (Fig. 75, 22). This pipe may be from 2 to 6 feet in length (according to the probable depth and importance of the hole); the bore of the pipe, say 4 inches, is just sufficient to take the largest chisel that may be used. The pipe has a flange at the top end. It is placed vertically in the hole in the centre of the pit, and the earth is firmly rammed round it; the flange rests either on the ground or on a plank with a hole in it, through which the pipe is passed. The pipe now forms a permanent entrance to the hole and a vertical guide for the chisel.

Above the pit is fixed the spring-pole (Fig. 75, 17). This is a young larch tree about 30 feet in length. The butt end is placed between two stout posts, which are securely planted in the ground; an iron pin about 1 inch in diameter passes through the posts and the pole, and so holds the pole down. At a distance, that may be varied from time to time, of say from 3 up to 8 feet from the butt end, a block of wood is placed under the pole so as to raise the end of it that is over the bore-hole to a convenient height—say 10 or 12 feet. There is also a pulley-frame. For shallow borings the frame consists of three legs lashed together (or pinned) at the top, from which a pulley ("snatch-block") is suspended. A windlass with a good hemp rope is also placed on one side and weighted down with stones. Fig. 75*a* shows a pulley frame, windlass, and balance-weight lever, sometimes used instead of a spring-pole.

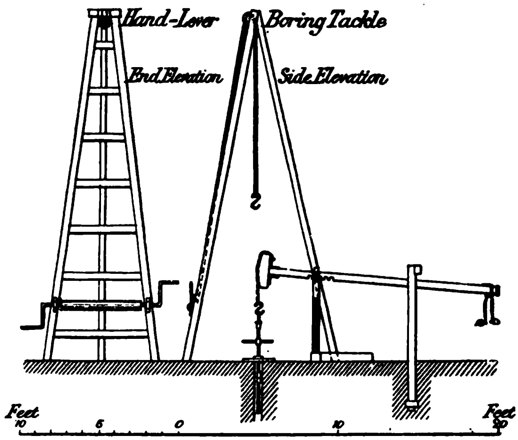


FIG. 75*a*.—Hand-boring frame and lever.

The boring is now begun. A couple of men take the chisel screwed on to the end of a rod, and, putting it through the pipe, strike it on the ground, turning it round between each blow; they also put some water in the hole to facilitate the cutting. When they have bored the hole about a half-yard they put on the cross-head, by means of which they can more easily lift and turn the rods; after boring another half-yard, they draw the rods, take off the short rods, and substitute a 6-foot rod, which with the chisel and cross-head make up a length of three yards; when the boring is another half-yard deeper, the half-yard rod is added at the top under the cross-head; after another half-yard is bored, the yard length is substituted for the half-yard length, and so on, the additions being so arranged as always to keep the cross-head at the most convenient height for the men to handle. The depth of the hole is always known by the length of rods. As soon as the length of rods gets too great for three or four men to lift with ease, the cross-head is suspended by a chain and swivel-hook from the end of the spring-pole. The weight of the rods bends the pole down; the chain is adjusted so that the pole holds the rods suspended just clear of the bottom of the hole. The men now press the chisel down to the bottom, and then lift it up assisted by the pole; then, pressing it down again, they strike a smart blow with the chisel, turning the cross-head between each blow, so that with every eight blows they make the chisel turn completely round.

After working for some time, varying from ten minutes to an hour, the bottom of the hole becomes choked with dirt or the chisel gets blunt; the rods are then drawn. This is done by means of the winch with a rope over the pulley. There is a claw-hook (Fig. 75, 20) at the end of the rope, which takes hold of the shoulder in the rod next below the cross-head piece. By this the rods are lifted, then unhooked from the spring-pole, and lowered an inch or two on to a fork (Fig. 75, 13) placed over the hole, which supports the rods by the lower shoulder. Then the cross-head is unscrewed by the key (Fig. 75, 14) and removed; the rods are now lifted as high as convenient, and again placed on the fork; the upper rods are unscrewed and reared up on one side of the pit or the pulley-frame. This operation is repeated until all the rods are drawn.

When the chisel is drawn it must be carefully examined, because adhering to it will be the *débris* from the bottom of the hole, which will show the nature of the stratum in which the chisel is working. When some small bits of this earth have been found and examined, it will be at once apparent whether the stratum is sandy or clayey, light or dark, coal or stone; whether it is hard or soft has been already ascertained by the men who were boring feeling the effect of each blow. Specimens of the stuff are to be carefully preserved and labelled.

The sludger (sometimes called the cleanser or sand-pump) (Fig. 75, 4) is now screwed on to the rods and lowered down the hole. The lengths of rod have now to be screwed together again as in turn they are put into the hole, until the sludger rests upon the bottom. The sludger is often auger-ended, and in that case it is turned round so as to screw into the sludge; it is also lifted up and down so as to force the semi-liquid sludge up through the valve in the bottom. The rods are then drawn again in the same way as when drawing the chisel. When the sludger reaches the top it is carried very carefully to a trough, into which the liquid contents are poured, the water being allowed to run off, and the sludge carefully examined, and some of the solid portions picked out, and specimens of the semi-solid parts put on one side to dry. The examination of the contents of the sludger is the most critical and important part of the operation, as by it mainly the nature of the ground can be told; and, the greater portion being in the form of sludge, the description of the ground has to be inferred, as it is obviously not identical with the pulverized material extracted. If the sludge is composed of sand, and if when boring the ground was found to be hard, it is evident that the stratum bored through was sandstone; if the sludge is clayey, and the boring was hard, it is evident that the stratum was a hard shale or a hard bind; if sand and clay are mixed in the sludge, the stratum is then a sandy bind or a sandy shale. The colour of the rock will be similar to the colour of the sludge, but there will also be fragments of the ground which will indicate its nature. The thickness of each stratum can generally be ascertained accurately, because, when the chisel passes from one stratum to another, some difference is felt by the man at the cross-head, and the depth of rods in the hole is then exactly noted. If the ground is not very hard the sludger will have to be sent down twice before the hole is clean. Then the chisel is sent down again; a sharp one is sent down each time.

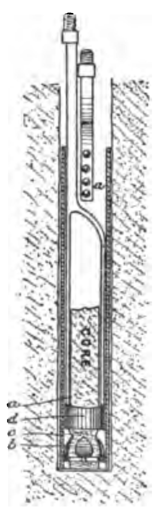


FIG. 76.—Core-grapnel.

It is often necessary to obtain larger specimens of the ground through which a boring is made than the small bits that adhere to the chisel or are drawn by the sludger, and for this purpose a core-cutter is used. This cutter often consists of a four-pronged fork. At the end of each prong is a chisel, as shown in Fig. 75 (15). This fork, instead of cutting the whole of the ground away, only cuts a ring, leaving a core standing up in the middle. A different kind of core-cutter, as shown in Fig. 75 (24), is sometimes used; this has more of a grinding action, and by twisting it round a circular groove is made.

The core having been cut, the next process is to break it off and lift it up, using the core-extractor, as shown in Fig. 75 (16). The bell *a* is lowered over the core *d*; the wedge *b* jams one side of the bell against the core, thus breaking it off; the springs *c* help to hold the core in the bell whilst it is being drawn up. Another core-extractor is shown in Fig. 76. The cylinder *a a* is lowered down over the core, the small steel-spring cutters *b* being forced back by the entering core; the cylinder is now turned round and slightly lowered and lifted alternately so as to cause the points of the four steel cutters to enter into the lower part of the core *c*. The iron ring *d* is lowered down inside the cylinder *a*, and over the core and on to the top of the four steel cutters, forcing them against the core. The cylinder *a* is now jerked upwards, and the core broken off and drawn to the top. In order to get a good core by this process it is necessary that the hole should not be too small, and it would probably be of little use trying to get a core with these tools in a hole much less than 4 inches in diameter. When a core can be got, it gives much more satisfactory evidence of the nature of the strata than can be obtained with the sludger.

Details of Apparatus for recording Direction and Angle of Dip.—In some cases the direction and the amount of dip of the strata can be ascertained from the core. Suppose, for instance, that when the core reaches the top of the hole it has not been turned since it was detached from the solid ground, then, if there are any lines of bedding in the sample, they may be noted, both as to direction and angle. But to avoid any error resulting from the turning of the core during extraction, it is sometimes marked before it is detached from the solid ground. A boring-tool, the lower end of which is a ring on which is fastened a cutter at the outer edge, is lowered into the hole. This cutter is very gently struck upon the top of the core, the position of the cutter as regards points of the compass being noted at the time the blows are struck. When the core is subsequently withdrawn, the marked side is placed in the same direction as that in which the cutter was previously held.

Speed of Borings.—When a hole is first started, with good men and good tackle, a boring in the coal measures, shale and bind, may be made at the rate of 12 inches an hour; but after a depth of some 20 yards is reached, the speed is much reduced, owing to the time required for drawing and lowering the rods. It is evident that the time thus occupied will increase with the depth of the hole, so that in deep bore-holes the progress is very slow, no matter how easy to bore may be the stratum, and with such apparatus as has been described the speed of boring would be quickly reduced to 12 inches in 24 hours.

Machine-Boring.—In order to increase the speed of boring, steam machinery is employed and other methods adopted to save time. If a high pulley-frame is used, a long length of rods can be unscrewed at once; thus if the pulley is about 70 feet above the platform, 60 feet of rods can be detached at once, and this causes a great economy of time. If a steam-winch or winding-engine is used for raising and lowering the rods, and if the sludger is lowered and raised by a rope instead of rods, there is a further economy of time; and if, instead of a spring-pole worked by men, a beam worked up and down by a steam-engine is used, a greater speed can be attained. With this improved apparatus the speed of boring can be quadrupled. Fig. 77 shows a boring-frame and machinery erected by M. Dru, a well-known engineer. A, boring-rods; S, screw by which they are lowered as the cross-head *h* is turned round; C, vertical steam-engine by which movement is given to the beam; E, lower stop-block used for giving a jar to the free-fall cutter; F, upper stop-block to prevent the rods descending too far; T, horizontal steam-engine for working the winding apparatus; V, drum, and V, rope for raising and lowering the sludger; Q, drum, O, chain, and P blocks, for raising and lowering the rods; M, housing to permit work to be carried on in winter. An apparatus of this kind is suitable for holes varying in diameter from 4 inches up to 4 feet.

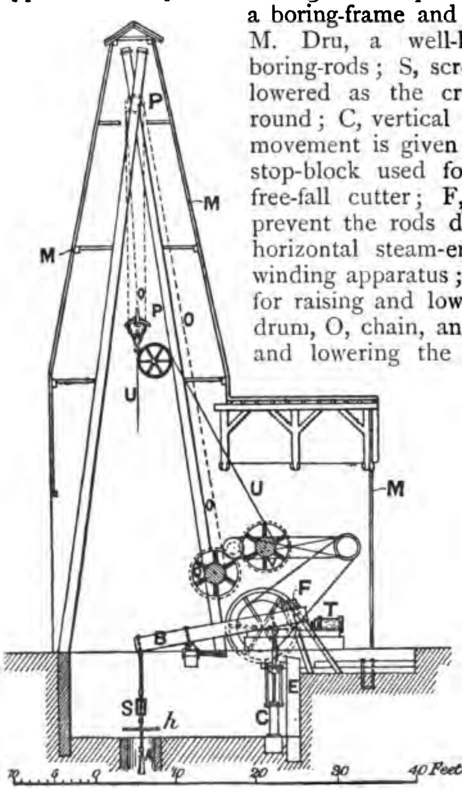


Fig. 77.—Vertical section of boring-shed.

In hard rock the progress by the old system of hand-boring is often exceedingly slow, perhaps not 1 inch in 24 hours; with steam machinery it is possible to strike much harder blows, and so obtain more rapid progress.

Fracture of Rods.—It is, however, at once apparent that if with a great length of rods, say 500 to 1000 feet, a rapid blow is struck, this will cause great vibration, a tendency to buckle, and a liability to fracture; indeed, this liability to fracture is so great that it is almost certain to occur before very long.

Sliding Joint.—To reduce this liability to fracture, a sliding joint (Fig. 78) is often used. This joint may be from 10 to 20 yards above the chisel, the length of rods B B below the joint being made specially heavy and strong. When the rods descend and the chisel strikes the ground, the upper length of rods, A, do not cease their movement, owing to the sliding joint, and they descend until the beam suspending them has completed its stroke, and thus the shock of contact with the rock at the bottom is avoided. On the upstroke the collar *a*, suspended by the fork F from the rods A, catches against the projecting cross-head M on B, and so the cutter is lifted again.

Free-fall Cutter.—Whilst the sliding shears save the rods, it is not altogether satisfactory, inasmuch as the speed with

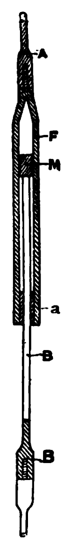
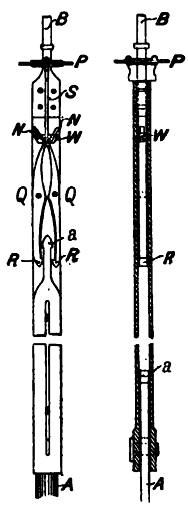
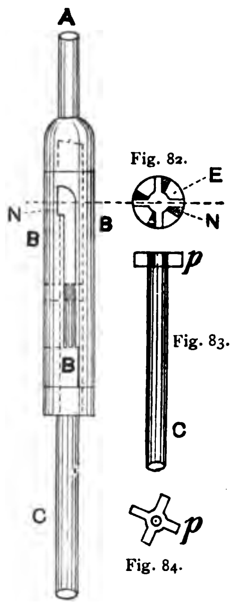


Fig. 78.—Sliding joint.



Figs. 79, 80.—Kind's disengaging apparatus for free-fall cutter.

Fig. 81.



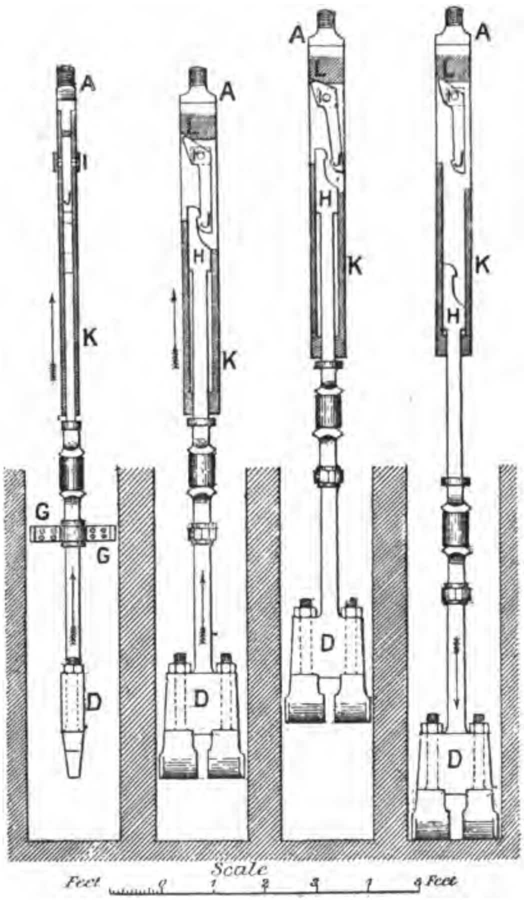
Figs. 81–84.—Fabian's disengaging apparatus for free-fall cutter.

which the chisel strikes the bottom is no greater than the speed of the whole mass of rods in the hole. To increase the speed and therefore the effectiveness of the blow, a free-fall cutter is used; with this apparatus the lower length of rods and the chisel are lifted from the bottom of the hole a suitable distance, say 2 feet more or less. They are then disengaged from the rods above and allowed to fall with the speed due to their gravity, modified by the resistance of the water. This speed is much greater than that with which it is safe to move the bulk of the rods. After the chisel has fallen, the rods are lowered to take hold of the chisel and then lift it up again. Figs. 79 and 80 show the disengaging apparatus as used by Herr Kind, a well-known German engineer. A A are the lower length of rods to which the chisel is fastened; *a* is the knob at the top; R R are the jaws of a pair of nippers which turn on the pins Q Q; B B are the upper rods fastened to the boring-beam; P is a piston rather smaller than the bore-hole, which is free to slide a short distance on the bore-rod B; W is a wedge attached to the piston by the thin rods S. Two holes are bored in the wedge at an angle of 40° from the vertical; through these two holes pass two rods N N, which are the upper levers of the nippers. If W is drawn up, N N are drawn together, and the jaws R R are opened, allowing the rods A A to fall. If W is lowered, N N are moved apart, and the jaws are closed, gripping the knob *a*. Suppose the rods in the bore-hole with the cutters suspended say 3 feet from the bottom, and the rods now rapidly lowered. The piston P, supported by the water, does not fall (except slowly); this draws the wedge W over the ends N, so opening the jaws and allowing the chisel to fall. The rods B B continue their descent until the jaws R R have come over the knob *a*. The rods B B are now moved upwards; the piston P does not move up on account of the resistance of the water and its own gravity, so the wedge W is forced down, so closing the jaws, gripping the little knob *a*, by which the lower rods and chisel are lifted.

Fabian.—Another engineer named Fabian has a free-fall cutter as shown in Figs. 81–84. As in Kind's apparatus, the lower length of rods is very strong and heavy, with a cutting tool attached to the bottom. The upper rods A terminate in a bell or cylinder B; the bottom of the cylinder is closed solid except for a hole E through which the lower rods C C can move; at the top of C is a cross-head *p* (Figs. 83 and 84). Four slots in the cylinder fit each head *p*; at the top of these slots they are widened as shown at N (Figs. 81 and 82); on the four seats thus formed by the enlargements the cross-head *p* can rest. Suppose the tool is suspended in the bore-hole, say 2 feet from the bottom, a sudden turn of the rods A from left to right will cause the cross-head *p* to slip off the seats at N and drop the length of the slot. The rods A are now lowered till the cross-head is at the top of the slot, when they are turned from right to left, catching under the cross-head *p*, which is thus lifted ready for another stroke.

Dru.—This disengaging apparatus (see Figs. 85–88) is used with a machine which is shown in Fig. 77. The upper rods A terminate in a cylinder or fork, K. At the top of this is a hinged hook, J, which catches on the knob H at the top of the falling rods. When the beam B (shown in Fig. 77) strikes the post E

Fig. 85. Fig. 86. Fig. 87. Fig. 88.



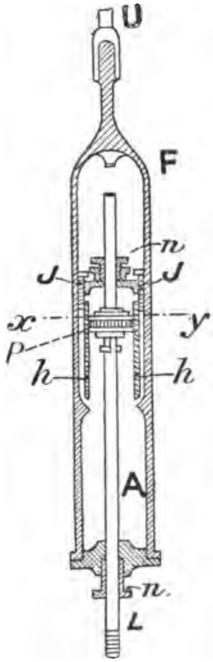
DISENGAGING APPARATUS FOR FREE-FALL CUTTER (M. DRU).

FIGS. 85 (side view), 86 (front view), boring-rod ascending and raising the tool. FIGS. 87 (at moment of shock), 88 (tool falling), liberation of tool by shock at top of stroke.

it causes a jar, and the cutting tool D (Fig. 87) jumps in the cylinder K, also the hook J, which has an enlarged eye and is thrown on one side by the inclined shoulder L (Fig. 87). The hook H is thus free to fall, as in Figs. 86–88; the rods A continue to descend till the hook J has re-engaged itself on the knob H, when it is again lifted.

Arrault.—M. Arrault, of Paris, has an hydraulic slide-joint (shown in Fig. 89). In this case the lower rods L have a piston, P, near the top; which works in a cylinder, A. The cylinder is suspended from the upper rods U by the fork F. Suppose the rods suspended in the hole with the chisel say 3 feet from the bottom, the weight of the lower rods and the cutting tool will cause them to fall. This they do at first slowly, because the cylinder is full of water, and the piston fits the upper end rather closely; the water is, however, able to escape through the holes *h h* which lead from the lower part of the contracted portion of the cylinder to the top of the cylinder, so that the water can pass through these holes from the under to the upper side of the piston; when the piston is past these holes it enters the enlarged portion of the cylinder, and thus can fall more rapidly. After the chisel has struck the blow, the rods U continue to fall until the contracted portion of the cylinder is over the piston; the upper rods are then lifted, raising the cutting tool at the same time, because the piston can only move slowly through the contracted part. The speed at which it can fall is regulated by the screws J J. *n n* are stuffing-boxes. Fig. 90 shows a section of the upper portion of the cylinder A.

Fig. 89.



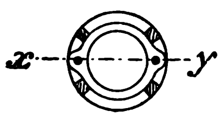


Fig. 90.

FIG. 89.—Free-fall cutter: hydraulic slide (Arrault). A, water-cylinder; P, piston working in cylinder; *n*, stuffing-boxes; *h*, holes through which water can escape; U, suspension-rods; J, screws to regulate flow of water.
FIG. 90.—Ditto. Section across *x y*.

Rods.—For deep borings it is common to substitute wood for iron in the construction of the rods. These wooden rods are made as long as the height of the pulley-frame permits, so as to reduce the number of joints, and will be in lengths of from 30 to 60 feet, and from 2½ to 6 inches square. At each end of the wooden rod an iron fork is bolted or riveted on; this fork is formed at the end into a rod with a screw, by means of which it can be joined to the next rod (see Fig. 91). In order to prevent the rods unscrewing if they are turned the reverse way, a locking-clamp is put over the joint (see Fig. 91). Since the wood is lighter than water, its buoyancy tends to support the weight of the iron joints, and thus the dead weight suspended in the hole and the strain upon the rods are reduced. In order to save time in raising and lowering the sludger, this is frequently done by means of a rope and steam winding-engine, the rods being only used for boring, core-drawing, and for lowering lining-tubes. To reduce the vibration of the rods in large holes, a lantern guide is used (see Fig. 92).

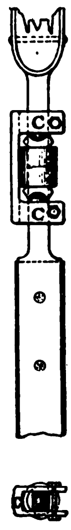


FIG. 91.—Locking clip to prevent unscrewing of couplings, scale $\frac{1}{10}$. *a*, plan of clip.

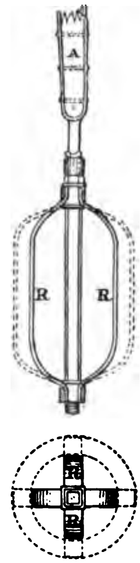
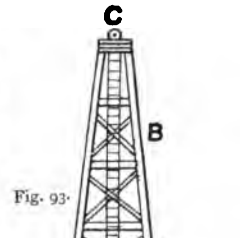


FIG. 92.—Lantern for guiding lower end of boring-rod. *a*, plan of lantern.

Rope Boring.—Instead of rods, a rope is sometimes used. Perhaps the best example of rope boring is the method practised in the Pennsylvanian oil regions, where bore-holes varying from 1000 to 2500 feet in depth are frequently put down. The apparatus used in this process is shown in Figs. 93–98.[1] The motive power is supplied by a steam-engine. The boiler is generally situated some distance away, say 50 yards or more, and the fire is frequently maintained by gas piped from some neighbouring bore-hole. A belt from the engine (Fig. 93, A) drives the machine at the top of the bore-hole. A length of rods 62 feet, weighing 2100 lbs. (Fig. 96, A) is attached to the end of a hemp rope and lowered to the bottom of the hole; the rope is then clamped to a cross-head, C (Fig. 96), suspended by chain and swivel from the end of the beam. The beam lifting the rope up and down causes the chisel to strike the bottom of the hole by its own weight, a sliding joint, called "jars," J K (Fig. 96), allowing the rope to descend after the tool has reached the bottom, thus keeping the rope straight. By means of the cross-head the chisel is twisted between each blow. As the hole is deepened, the temper-screw B (Fig. 96) is untwisted, so lowering the cross-head and clamp D to which the rope is attached. When this screw has been run out, the clamp is

[1] Figs. 93-98 are taken from the report of the Second Geological Survey of Pennsylvania.

slackened, the screw is run up again, and the clamp tightened again 3 or 4 feet higher up the rope. The tool is withdrawn by the drum D (Figs. 93–95), driven by the steam-engine; and the sludger is attached to the end of the sand-reel line. The sludger is here called a sand-pump, and contains a pump-bucket as well as a clack, somewhat similar in principle to that shown in Fig. 101. A few strokes of the bucket fill the shell with sand or sludge, which is then withdrawn; this operation may be repeated several times to clean the hole. The work is carried



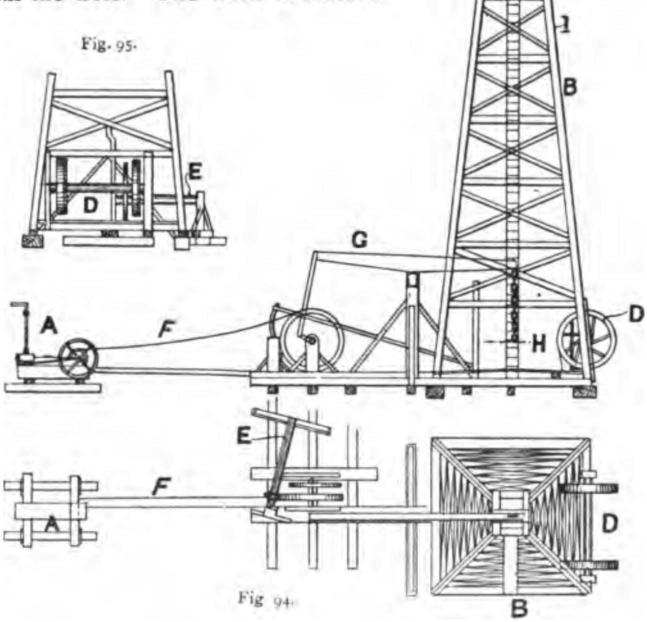


FIG. 93.—American rope-boring: side elevation. A, engine; B, pulley frame; C, pulley; D, Bull-wheel; E, sand-line reel; F, driving belt; G, Boring-beam; H, cross-head; I, rope. FIG. 94.—Horizontal projection. FIG. 95.—End elevation.

on by two men, who also regulate the engine by long rods, and sharpen the tools; if the bore-hole is going day and night, they are relieved by two other men. In ground which is easy to bore, a great speed is attained, varying from 20 feet to upwards of 200 feet in twenty-four hours. The height of the derrick is 75 feet from the ground. The method of lining the holes with iron tubes is shown in Fig. 97.

In this case the upper part of the hole is lined with 8-inch pipes driven through the soft ground into the bed of rock below. The hole is then bored $7\frac{7}{8}$ inches in diameter, and down this are put pipes $5\frac{5}{8}$ inches internal diameter. The bottom of these pipes has a sharp cutting edge, which, when driven fast into the strata, may make a water-tight joint, so that the water met with in the upper part of the boring will not find its way into the hole. In

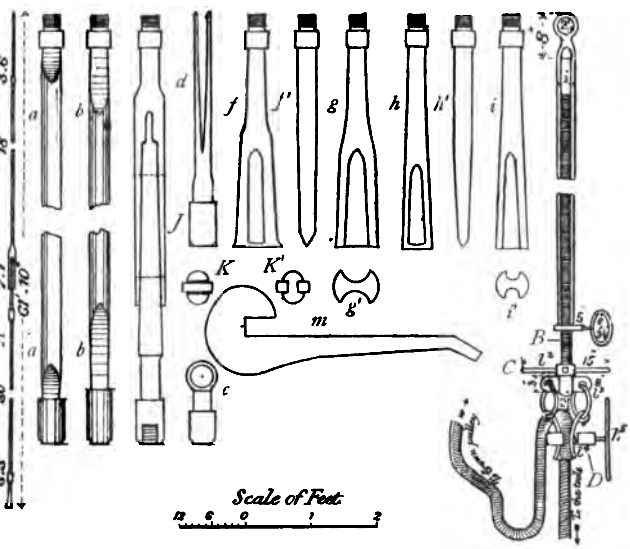
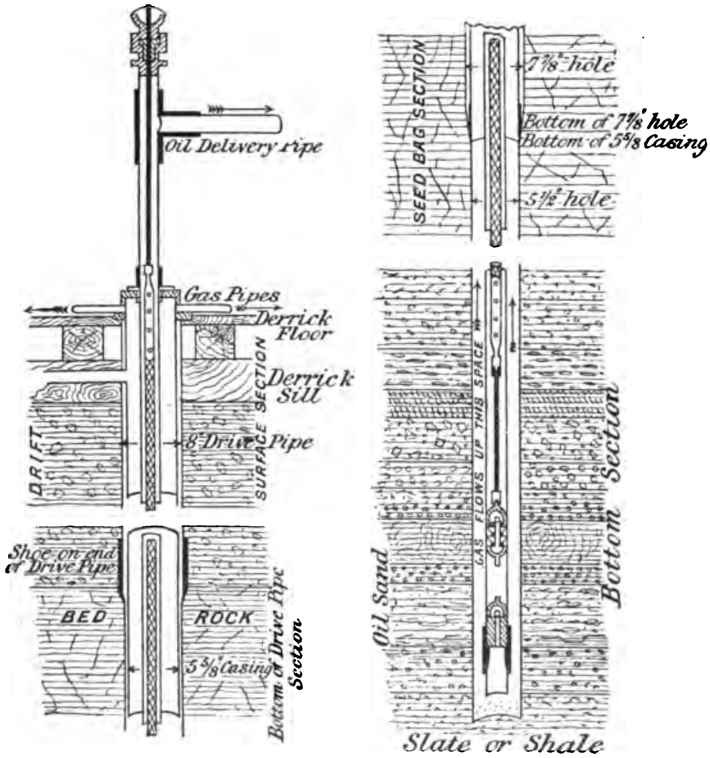


FIG. 96.—Tools used in drilling oil-wells. *A*, rods and chisel; *B*, temper screw; *C*, cross head; *D*, rope clamp; *J*, jars; *K*, *K'*, section of jars, *J*; *a*, sinker bar; *b*, auger stem; *d*, rope socket; *e*, ring socket; *f*, club bit; *f'*, edge view same; *g*, reamer; *g'*, bottom view same; *h*, centre bit; *h'*, side view same; *i*, reamer; *i'*, bottom view same; *m*, winch.

order, however, to make sure, the $5\frac{5}{8}$-inch tube is surrounded near the bottom with a bag several feet long filled with flax seeds, this bag being lowered with the tube to which it is tied fast at the top and bottom. After a time the flax seeds swell and compress the bag against the ground and the pipes, thus helping to make a water-tight joint. A third tube of 2 to 3 inches bore is lowered nearly to the bottom of the hole. At the surface a cover is fixed on to the $5\frac{5}{8}$-inch pipe, and the small pipe passes through a gas-tight joint. Gas can pass up between the small pipe and the $5\frac{5}{8}$-inch pipe, and can be conveyed away to be burnt by branch tubes; the centre pipe, which is the oil-pipe, has also a delivery branch. When first the hole is bored, the pressure of the gas forces the oil out through the small tube, and it is conveyed by pipes to tanks. As the supply of oil gets less, it is often necessary to put down pumps, as shown in the figure. A small clack-valve is placed at the bottom of the internal tube, a small bucket works above it, the top of the bucket-rod works



FIGS. 97, 98.—American rope-boring: lining of holes.

through a stuffing-box at the top of the tube, and the oil is pumped up. When the supply of oil gets still shorter, the bottom of the hole is sometimes enlarged by means of a torpedo. This consists of a long tin tube containing say 40 quarts of nitro-glycerine, more or less; at the top of the tube is a large percussion cap and hammer. The tube is lowered into the hole by a cord, and when it has reached the bottom an iron weight is dropped down the shaft, which hits the hammer and so explodes the torpedo. This has the effect of enlarging the bore-hole at the bottom and increasing the flow of oil to the hole.

Flat Ropes.—In England and on the Continent flat ropes have been used for boring, and have some advantages. One of the best-known methods of boring with flat ropes is that of Messrs. Mather and Platt. Their boring apparatus will be understood on reference to Figs. 99 and 100. The steam is brought from some boiler to operate two separate engines; one of them, D, is

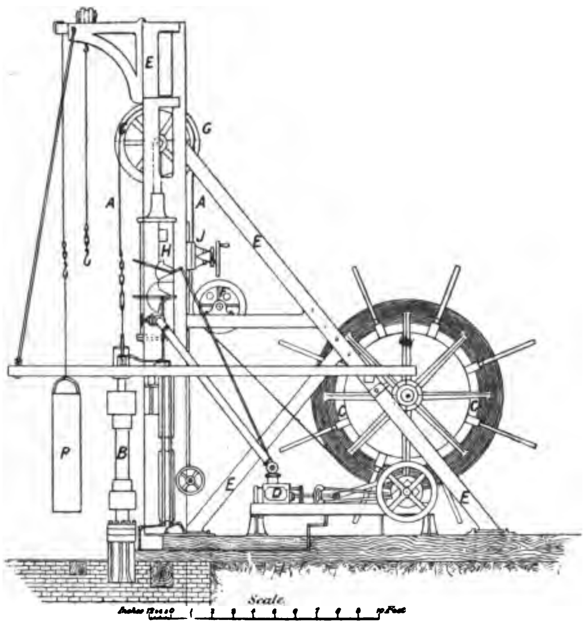


FIG. 99.—Mather and Platt: engines and frame.

the winding-engine, by means of which tools are raised and lowered; the other, H, is the boring-engine, by means of which the rope is raised a sufficient height for each stroke of the boring-tool. The tool B and also Fig. 100 is a heavy steel or iron bar; at the bottom are fixed some steel chisels, the number depending on the size of the hole, say eight or more. When about to work the tool, which has been lowered to the bottom of the hole, the clamp J is fastened, holding that end of the rope tight; steam is then turned into the boring-cylinder, which raises the piston-rod, carrying above it the pulley G. It is evident that, if this pulley is raised 1 foot, the tool suspended in the hole must be raised 2 feet, because the rope on the other side is made fast by the clamp; by opening a valve the steam escapes from under the piston, which then drops, and the tool, which falls twice as fast as the piston, strikes a smart blow at the bottom of the hole, the hardness of the blow being regulated by the height to which the tool is lifted, the engine-driver being able to regulate this by special

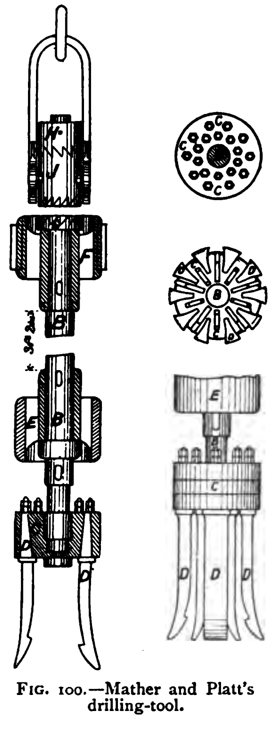


FIG. 100.—Mather and Platt's drilling-tool.

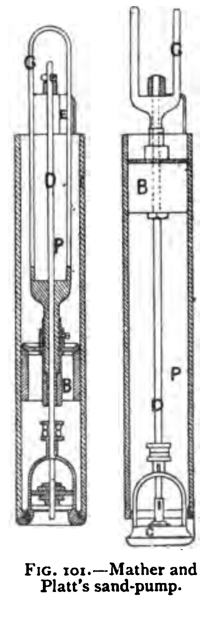


FIG. 101.—Mather and Platt's sand-pump.

valve-gearing (the valves work automatically). An ingenious device is employed for twisting the tool between each stroke; this will be understood on reference to Fig. 100. The sliding collar J has a toothed edge top and bottom. On raising the rope this sliding collar fits into a fixed collar at the top of the boring-bar, the lower edge of which has a corresponding set of teeth; on lowering the rope when the tool strikes the bottom, the sliding collar descends, and the lower set of teeth on the lower fixed collar become engaged, but in order that they may fit together the sliding collar receives a slight twist; upon raising the collar the upper teeth now engage in the teeth above, but not in the same teeth as before, the collar having turned one tooth, thus putting a slight twist upon the rope. When the weight of the tool is lifted from the ground, the tension due to this weight tends to untwist the rope and slightly to turn the tool as it is held in suspension, so that when it strikes the bottom of the hole it cannot strike in precisely the same place as before. To cleanse the hole the sand-pump is used (Fig. 101) in a similar manner to that already described in round-rope boring. With this machine holes varying in diameter from 8 inches up to 2 feet have been bored,

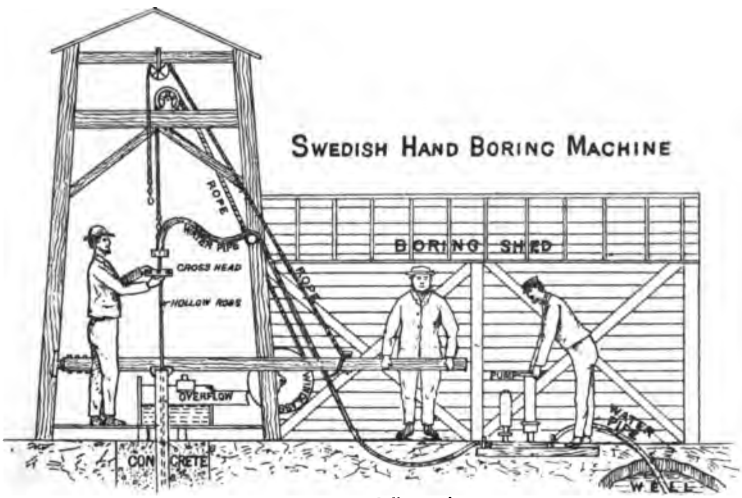


FIG. 102.—Hollow rods.

with varying success. It would appear, however, that in hard ground there is some liability to fracture.

Hollow Rods.—In place of solid rods, hollow rods made of steam tubes screwed together are sometimes used, the chisel being attached rigidly to the rods, down which a stream of water is pumped (Fig. 102). The water, issuing from the rods just above the cutting chisel, returns outside them, and carries with it the sludge and gravel made by the chisel; the sludge is deposited in a can, from which the water flows at the top, and by continual observation of the deposit brought up by the water the nature of the strata can be gathered. By this method of boring it is only necessary to draw the chisel when it is blunt. It is not, however, largely practised, and it is only suitable for soft ground.

Broken Rods.—In case the rods should break, a grapnel is used to extract the portion remaining in the hole. Fig. 75 (9) shows a simple kind of grapnel used for the ordinary iron rods. It is a tube about 5 feet long, open and bell-shaped at the bottom, with a screw at the top by which it is attached to the rods; inside the tube are four steel blades which spring from the sides near the bottom. To extract the broken rods this tube is lowered until it passes over the broken end of the rod, which rises up inside the tube until the broken end has reached the top; the tube or grapnel is then raised until the steel blades catch against the shoulder on the broken rod, beyond which they will not pass, and the tube cannot now be withdrawn from the hole without either a fracture of these steel blades or else raising the broken rod. There are many forms of grapnel.

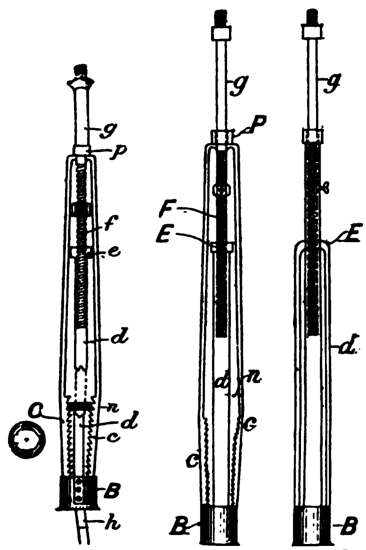


FIG. 103.—Grapnel jaws.

Wolff's Grapnel.—This grapnel is designed to catch hold of a broken rod, but is so constructed that, if it is found impossible to withdraw the rods which it has seized, they may be unfastened, and thus the upper length of rods can be withdrawn, which otherwise would be impossible if they were held fast by the grapnel to the broken rods below, which, in their turn, were jammed fast in the hole. Fig. 103 shows this grapnel.[1] B B is a bell-mouth suspended by the fork *d d* and the nut *e* from the screw *f*, the thread of which is turned in the lowest boring-rod; *c c* are steel jaws with sharp teeth slanting up, kept apart by the piece of wood *n*, and suspended from the rod *g* by the collar P. When B is lowered over the end of a broken rod, *h*, the wood is pushed out by the rod, and the jaws spring together upon the rod, and if the upper rods are now turned round, the jaws and the bell are prevented from turning, consequently the nut *e* is

[1] J. C. Jefferson, Midland Institute of Mining Engineers.

lifted and the bell B is drawn up and compresses the jaws, so that they hold the rod firmly. In order to relax this grip the rods must be turned the other way; then the nut *e* will be lowered and the bell with it. Other tools for recovering broken rods are shown in Fig. 104, and in Fig. 105 is shown the application of the hook-grappler.

Parachutes.—In boring deep holes it sometimes happens that, in withdrawing the rods, by some accident they are allowed to fall; if they had to fall a great distance before striking the bottom, the result would be a disastrous smash. Parachutes are therefore placed at intervals on the rod. These are pistons loose on the rods, sliding between two collars; they do not fit tight in the bore-hole, but leave space for the water to pass by them as they are raised up and down. When the bore-rods are raised, these pistons are supported by the bottom collar; when the boring-beam goes down and the rods are lowered for the blow, they slide through the piston faster than it can fall with them, owing to the resistance of the water; thus there is not much resistance to the working of the rods on account of these safety-pistons. If, on the other hand, the rods should be accidentally dropped, the pistons, catching against the upper collar and bearing against the water in the hole, limit the velocity of descent, and thus reduce the chances of breakage.

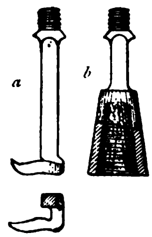


FIG. 104.—Tools for recovering broken boring-rods, etc. *a*, claw; *b*, screwed socket.

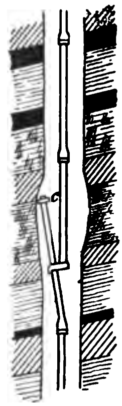
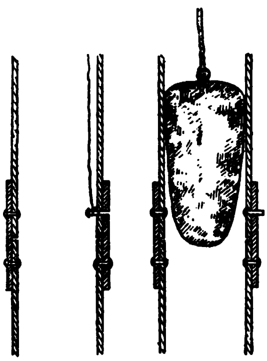


FIG. 105.—Application of hook-grappler.

Lining-Tubes.—When a hole is bored through soft ground the sides of the hole are apt to break off in places and encumber the bottom of the hole with *débris*. This is due partly to the rods occasionally knocking against the sides, partly to the wash of the water, and partly to the pressure of the ground itself upon a soft stratum, whether this be sand, gravel, or clay. To prevent this falling-in of the sides, it is necessary to line the hole with tubes; these tubes have been made of wood, copper, zinc, cast iron, and wrought iron, and perhaps of steel. Wrought-iron tubes are those that have hitherto been chiefly adopted; these are sometimes made with a longitudinal seam riveted, but for small holes a welded seam is now generally used; the joints at the end of each tube are sometimes riveted and sometimes screwed (see Fig. 75). When riveted joints are used, the rivet is lowered from the top of the last tube by means of a wire (see Fig. 106), and by means of a hook is pulled through the rivet-hole. When the rivets have been all placed in position, a metal block somewhat tapered (see Fig. 107) is lowered down inside the tube until it presses tight against all the rivet-heads inside, which are then riveted over cold on the outside; these riveted joints are sometimes slightly tapered (see Fig. 75, 8*a*), so avoiding the use of a collar. Sometimes an outside collar is used (see Fig. 75, 6), and sometimes a flush-joint is preferred (see Fig. 75, 7). Where the joints are screwed, there is sometimes an outside collar (see Fig. 75, 8), and sometimes there is a flush-joint (see Fig. 75, 5); where there is a flush-joint the middle of the tube must be thick enough to have nearly half turned down before it is threaded, and therefore it requires to be nearly twice as thick as where there is an outside collar. This system of flush-jointed tubes is therefore more expensive than the other, but it is often more convenient, because, in lowering the tubes down the hole, there are no external projections to catch against the sides of the hole, and in case the tubes should have to be withdrawn—an operation which is sometimes done—there are no projections to hinder.

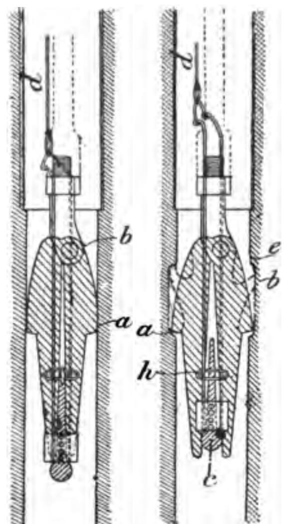


FIGS. 106, 107.—Method of riveting tubes.

Reduction in Diameter of Hole.—It is evident that, if a lining-tube has been placed in the hole, the next boring-tool introduced must be smaller than the original one, and this reduction in diameter can hardly be less than $\frac{3}{4}$ inch, while it may sometimes amount to $1\frac{1}{2}$ inch; therefore the introduction of two or three lining-tubes will reduce a hole of moderate size to a diameter which is too small for practical purposes. For this reason, if the hole has to go a great depth, it must be started of such a diameter that numerous linings may be introduced, and still leave a sufficient diameter for further boring; thus if a hole was started 12 inches in diameter, six lining-tubes might be introduced, and still leave a hole 6 inches in diameter at the bottom. And very deep holes, that is to say, holes of 2000 feet and upwards, should not be started less than about 18 inches in diameter. Of course, if it is known at the outset that sands and very soft clays will be met with near the surface, a still larger diameter might be adopted.

It frequently happens that the necessary number of lining-tubes has not been foreseen, and that the hole has been already reduced so much that it cannot be carried further if additional lining-tubes are introduced, and it becomes desirable to drive the last set of lining-tubes further down the hole. To make this possible, it is necessary to rime out the hole. Numerous kinds of rimers have been made, some consisting of curved chisels with sharp sides opening by means of a spring.

The kind of rimer made by Kind is shown in Fig. 108. In this case the cutter is made of two pieces of steel hinged on the pin *b*, with shoulders, *a*, on which is formed the cutting edge. The cutters can be closed together as shown in Fig. 108, or they can be opened out as shown in Fig. 109. This is effected by means of the wedge *c*, which is drawn up by the cord *d*, opening out the chisels to the distance permitted by the link *h*. By means of this or similar tools the hole may be enlarged.



FIGS. 108, 109.—Widening borers cutting down and cutting up.

It is evident that this form of chisel leaves a shoulder of rock, *e*, supporting the tubes. In order to remove this shoulder, a similar tool to that shown in the figure is used. But the cutting edge *b* is formed to cut upwards, as shown by the dotted lines in Fig. 109, so that by jerking it upwards the shoulder *e* is cut away.

Diamond Drill. — The diamond drill is an example of the process of boring by grinding instead of percussion. The diamond is the hardest or one of the hardest substances known, and is much harder than any rocks through which borings have to be made, and therefore, if a diamond is pressed against any other substance and then moved along the surface of this substance, it will scratch it. There is a species of diamond which is of no value for ornamental purposes, and is therefore much cheaper than the brilliant; it is this dull kind of diamond which is used for boring. Fig. 110 shows the apparatus. Fig. 111 is the boring head or crown, which may vary from 2 inches up to 18 inches in diameter. This is a ring of steel, with shallow holes dovetailed as shown by the black marks; into these holes the diamonds are placed surrounded by some soft metal, and then the metal is hammered over them, completely enclosing the stone, all excepting one slightly projecting point or corner. This boring-crown is screwed into the end of a long length of

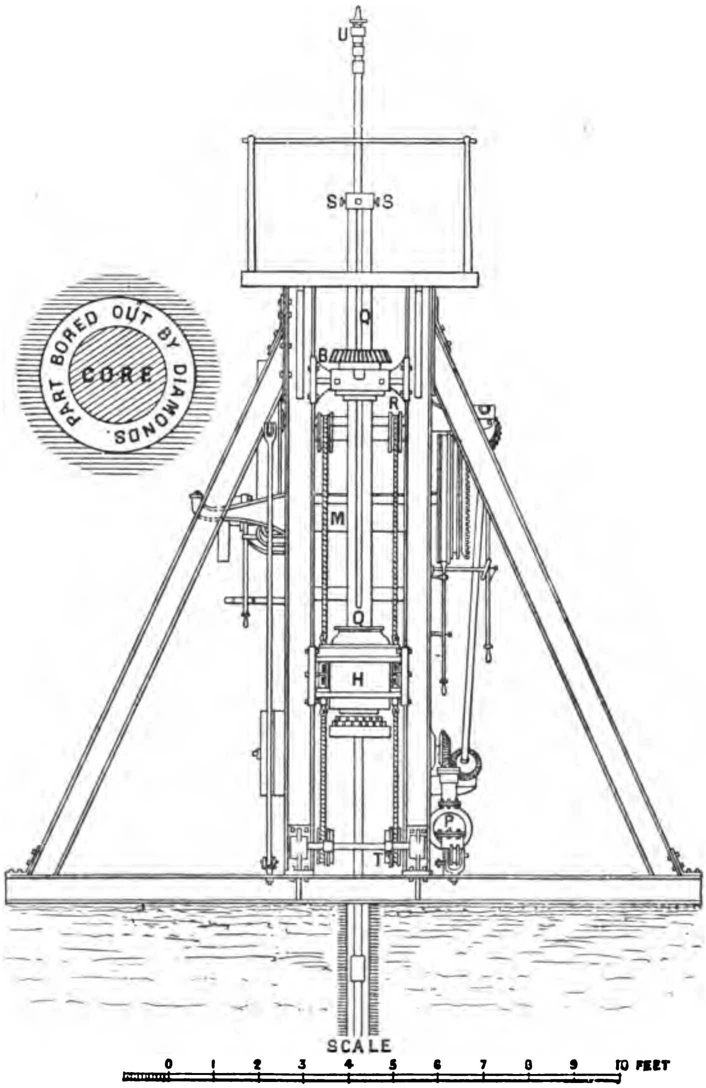


FIG. 110.—Diamond drill machine, front elevation.

steel tube (Fig. 112). At the surface the tubes are gripped in a tube, Q Q (Fig. 110), which has on the outside a rib or feather; this feather slides vertically in a collar, which is fastened to and made to revolve by the bevelled gearing B. The upper part of the boring-tube is connected by water-tight and swivel joints U with a hose-pipe connected with a force-pump, P; by means of this pump water is forced into the interior of the tube, and down to the bottom of the hole, where it issues underneath the boring-crown. When the engine is started, the boring-rods revolve at a speed of perhaps 200 or 300 revolutions a minute, and as they are a great weight they force the diamonds against the rocks, which are abraded; at the same time the water, issuing from the internal tube, carries away the sand that is made, keeping a clean surface against which the diamonds can act. The tubes are free to slip down inside the outer collar, and thus are continually pressing at the bottom of the hole as it becomes deeper. When the tubes have fallen the length of the outer collar, the gripping-tube is unclamped from the boring-rods and raised up to the top of the outer collar, when it is again clamped, and fresh rods are added at the top as the bore-hole gets deeper. As soon as the bore-hole gets a considerable depth, the weight of the rods becomes too great to rest upon the diamonds without causing too much friction; this weight is reduced by balance-weights. Wire ropes or chains from the balance-weights pass over pulleys R, and are connected with a loose collar, H, fixed on the gripping-tube, within which it is free to revolve, but by which the weight of the rods is supported. The deeper the hole, the greater the balance-weight required.

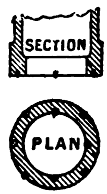


FIG. 111.—Boring-crown.

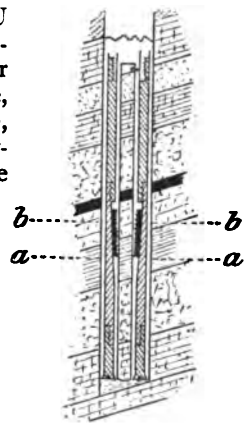


FIG. 112.—Core-extractor.

Cores.—As the boring-crown only cuts a ring, the centre part remains and forms a core, which passes up inside the hollow rods. The lower part of the rods are called a core-tube, and this tube is sometimes of sufficient length to admit a core of 20 feet, and is usually of larger diameter than the rods above; but in order that the core may be withdrawn at the same time with the rods, it is necessary that there should be some kind of grip or grapnel inside the core-tube. This is made in the following manner: Three vertical grooves at opposite sides of the core-tube are cut with an incline so that the depth of the groove is less at the bottom of it than at the top, *a a* (Fig. 112). Into each of these grooves is placed a sliding block of steel, *b b*, with serrated edges inside; these blocks press against the core, and are lifted

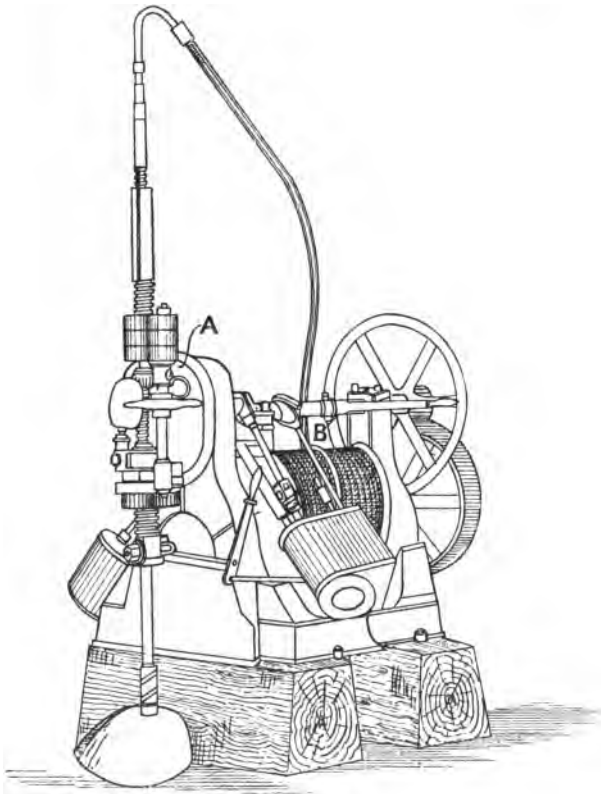


FIG. 113.—Diamond boring-drill.

up by it, as it enters, towards the top of the groove, where, owing to the greater depth of the groove, there is room for the core to pass. If now the core-tube is raised, and the movable blocks, catching against the core, remain stationary, the inclined surface of the groove will press them so tightly against the core, that by the time the bottom of the groove comes against them and forces them up, they will also break off the core from the bottom, and bring it up inside the tube (a split ring is sometimes used instead of separate pieces).

This method of boring is particularly applicable to hard rocks, and is nowadays generally employed in such ground. In rather tender ground, such as is common in the coal measures and in the New Red Sandstone formation, borings have not always been successful, because the core has been shattered by the shaking of the drill, and then carried away in the form of sludge; the nature of the sludge has escaped detection, and a bore-hole has been known to pass through a good seam of coal, unknown to those in charge. In order to get good cores in the coal measures, the hole should not be less than 6 inches in diameter, and must therefore be started at a still larger diameter. A special core-holder is made to protect the core, and has been used with good results.

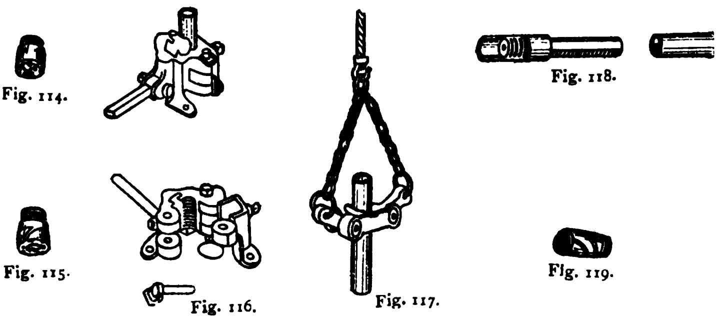


Fig. 114. Fig. 115. Fig. 116. Fig. 117. Fig. 118. Fig. 119.

FIG. 114.—Boring head. FIG. 115.—Core bit. FIG. 116.—Safety-clamp. FIG. 117.—Lifting-jack. FIG. 118.—Bayonet clutch coupling. FIG. 119.—Core-lifter.

This core-holder has an internal tube round which the boring crown revolves, the internal tube remaining stationary; the water from the boring-rods passes down between the internal and the external tubes; the core is thus saved from direct contact with the revolving tube and the stream of water. At the top of the internal tube are some holes through which the water escapes as the core rises up.

A similar machine (see Fig. 113) is used in the ironstone-mines of Michigan, U.S.A., for prospecting. This machine is taken into the mines, and bores a hole through the rock at any angle that may be desired, vertical or inclined, either at an angle of 45° or level. The head A (Fig. 113) revolves, and can be clamped at the desired angle. On the machine is a winding-drum, B, by which the rods can be raised or lowered. Fig. 114 is a boring-crown, which grinds all the stone away. Fig. 115 is a core-cutter. Fig. 116 is a clamp by which the rods are held in the hole, when being jointed or unjointed. Fig. 117 is the clamp-hook with which they are lifted. Fig. 118 shows the joint by which the rods are attached one to the other. This joint saves the time required for screwing and unscrewing, one quarter-turn being sufficient to make a fastening. Fig. 119 is a core-lifter.

Davis Calyx Drill.—The Davis calyx drill has lately come to the front, having been much used in Australia, and more recently in England. It is similar in principle to the diamond drill, but instead of a boring crown set with diamonds, the crown is shod with steel teeth, set a little in and a little out alternately, as in a saw, so as to clear a little space between the core tube and the core and the sides of the hole.

Above the core-tube is another tube surrounding the hollow rods and rather less in diameter than the hole; this tube is any convenient length—say 10 feet or more. It is closed at the bottom and open at the top. This tube is the "calyx," or cup. The water forced down the hollow rods, rising up the hole in the narrow annular space between the core-tube and the calyx and the side of the hole, carries upwards the sand and gravel made by the boring crown, but on reaching the top of the calyx the water-space is enlarged, and consequently the speed of the current diminished; the gravel therefore falls into the calyx. The calyx thus catches samples of the strata, and also reduces the liability to clogging by falling fragments. This drill will bore through shales and sandstones, and gives very good cores.

Choice of Method.—In deciding upon the system of boring to be adopted, the purpose of the hole has to be considered. For exploration purposes it is very important to get good cores. For the purpose of a well, whether for oil or water, the cores do not matter; it is only necessary to get the hole down quickly and straight. In many exploring holes cores are required of the whole of the boring; and in many others only of a few places, the position of which is known approximately beforehand.

Cost of Boring.—With regard to the cost of boring, this varies so greatly with the nature of the ground and the wages and skill of the workmen, that it is difficult to say beforehand. There is also the element of profit and loss on the part of the contractor. When boring by hand, with a spring-pole or a balance-beam, such prices as the following are sometimes paid in ordinary coal measures: For a hole starting at 3 inches in diameter, the contractor finding his own boring-rods and tools, but not the spring-pole or windlass: for the first 5 yards, 5*s.* 6*d.* per yard; for the second 5 yards, 7*s.* 6*d.* per yard; for the third 5 yards, 9*s.* 6*d.* per yard; and so on, rising 2*s.* per yard every 5 yards.

Some of the figure

	Rigid rods.		
berg.	Malkowitz.	Sperenberg.	
		Hand-power. ft. in.	Steam powe… ft. i…
·8 in. ·3 in.	3 ft. 4·1 in. 1 ft. 10·2 in.	3 5·3 1 10·8	7 0… 2 6…
		2 ft. 9 in.	
	2 ft. 4·2 in.		
n and oni- strata	1000 ft. Permian, the rest Carboniferous and Silurian strata	Gypsum, anhy- drite, and rock salt	
	6 ft. at a depth of 1400 ft.	—	
o	1857	4170	
a.	24 in. 7 in. —	— —	
		Hand-power.	Steam power
9 8	4896 3912	3286 2464	5275 9563
7	8808	20,588	
	734	1713	
	48*s*. 4*d*.	41*s*.	
12	£4489 8*s*.	£8567 15*s*.	

A simple rule for calculating the average cost per yard is, calling D the depth in yards, and x the average cost per yard—

$$\frac{D - 5}{2 \times 5} \times 2 + 5 = x$$

equals cost per yard in shillings; and the total cost = $x \times D$.

In the case, therefore, of a hole 105 yards deep, the cost is found as follows:—

$$\frac{105 - 5}{2 \times 5} \times 2 + 5 = 25$$

the cost per yard in shillings, and the total cost = 105×25 = £131 5*s.*

If the hole is 305 yards in depth—

$$\frac{305 - 5}{2 \times 5} \times 2 + 5 = 65s.$$

total cost, $305 \times 65s.$ = £991 5*s.*

It thus appears that a hole 305 yards deep is about eight times as costly as one 105 yards deep, and costs, in fact, rather more than £1 a foot. In various districts the price may be a little more or a little less. In the American oil-wells, the holes are put down very cheaply; but then very few samples are taken of the strata bored through, and the cost per foot is, for labour, only from 2*s.* a foot as a minimum up to 4*s.* a foot as a common figure, and four or five times that amount in some cases. The cost per foot in the American copper and ironstone districts is also very moderate. With a diamond drill giving a core $1\frac{1}{8}$ inch in diameter, and for distances not exceeding 300 or 400 feet, it is about 5*s.* 6*d.* a foot, including all costs except steam-power. The costs of some bore-holes on the Continent are given by Mr. J. Clark Jefferson, in a valuable paper read before the Midland Institute of Mining Engineers, in 1878.

Table VI. shows a summary of the cost of some bore-holes in Europe and America.

It is probable that diamond boring can now be done at a much less cost than in many of the earlier cases, great experience having been attained in the manipulation of the tools, and in the improvement of the hollow rods, core-tube, etc.; and holes can probably be put down at the following prices: surface to 1000 feet, say £1 per foot; 1000 to 1500 feet, £1 10*s.* per foot; 1500 to 2000 feet, £2 10*s.*, per foot; 2000 to 2500 feet, £5 per foot. These prices would include the extraction of cores.

CHAPTER IV.

WINNING; OR, MODE OF APPROACHING THE MINERAL.

After exploring an estate, and having decided to proceed to get the minerals, the next question is to choose the best method. This, perhaps, will be most clearly explained by considering a few examples.

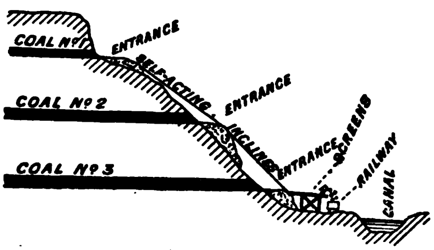


Fig. 120.—Winning level seams from hillside.

Fig. 120 shows a very simple case of frequent occurrence—several seams of coal lying in horizontal beds, and cropping out on the side of a hill. A drift or heading is made straight into the coal, and continued, as required, across the estate; the mineral from the workings is lowered down to the railway or canal at the bottom of the hill by means of self-acting inclines.

Fig. 121 shows a seam of coal inclined, cropping out near the top of a hillside. A heading is made into the coal, going downhill in the seam. This, however, is soon stopped by water, and a level drift is driven in from the lower part of the hillside, just above the river, across the measures, until it reaches the seam of coal; all that part of the seam which lies above this level drift is thus drained of water.

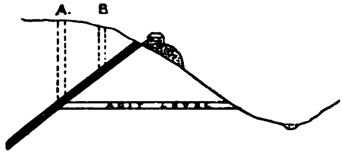


Fig. 121.—Winning inclined seam by adit.

A level drift is sometimes called an "adit," or "sough." For the purpose of ventilation a shaft is sunk at B, and subsequently at A. The coal is drawn to the surface, up an incline, by an engine placed on the hillside.

Fig. 122 represents two seams of coal dipping under a hill at a moderate inclination, and cropping out, one in the valley, and the other on the hillside. Two shallow pits have been sunk to the upper seam, a pumping-engine being used to drain the coal. All the coal above the level where these shafts cut the seam having been exhausted, an incline drift is driven from

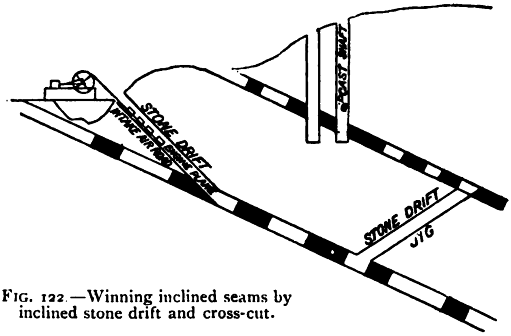


FIG. 122.—Winning inclined seams by inclined stone drift and cross-cut.

the hillside down into the lower seam, which is found to be dry, and headings in the coal are driven to the dip, to the boundary of the estate, or as far as required. A cross-measure drift is driven up into the upper seam, and headings are thence driven in the coal. The mineral is lowered down this cross-measure

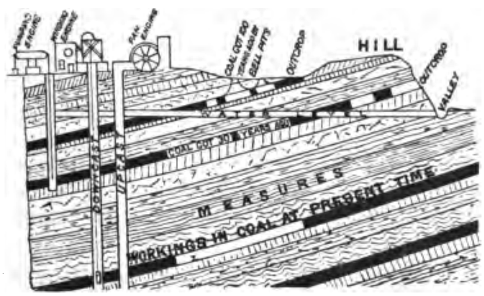


FIG. 123.—Section of coal-mines. Winning seams of moderate inclination by shafts.

drift by a self-acting incline, called a "jig," and is drawn up an engine-plane in the lower seam and stone drift to the surface.

Fig. 123 shows four seams of coal lying at a moderate inclination, two of them cropping out just above a little brook, on ground which rises to a height of 100 feet above the brook. Near the outcrop the coals have been partly worked by shallow pits; on the "dip" side of the estate three shafts have been sunk to these upper seams.

In order to let off the water, of which there is a large supply, a water-level is driven from the valley, at an inclination of about 1 in 600, to the pits, and all the water made above this level is thus drained.

In order to reach the lower seams, it is not necessary to deepen the pumping-pit, as much water is not expected below the second seam. The water from the bottom of this pit is pumped up to the water-level. The winding-pit and the upcast, or fan-pit, are sunk down to the third seam, and, at a later date, they will be continued down to work the fourth and other seams still deeper.

Fig. 124 shows five seams of coal at a steep inclination, perhaps 12° to 50° from the horizontal. Two shafts are sunk vertically to such a depth as is convenient and attainable at a reasonable expense—say from 100 to 500 yards, as the case may be. From near the bottom of these shafts a stone head or cross-cut is driven level in both directions, cutting across all the five seams. Headings are then driven on the level in three of these seams, which happen to produce the most marketable coal, and from these coal-levels workings are driven to the rise and, in some cases, to the dip.

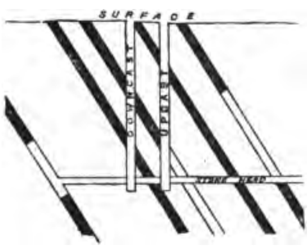


FIG. 124.—Winning seams of steep inclination by shafts and cross-cut.

With the exception of the cases of "open holes" and "bell pits," referred to in another chapter, all stratified seams can be won by one of the above five methods.

Fig. 125 refers to a metalliferous vein, which traverses the ground, dipping at an inclination of say 70° from the horizontal. It crops out at the surface, and a pit, or shaft, is excavated in the vein, and is carried down deeper and deeper as long as the working of the vein is profitable, levels being started out of the shaft about every 20 yards and driven in the vein; an adit is driven from the valley to cut the vein, and so let off the water. In some countries adits are of great length—upwards of 12 miles. In the case of extensive mines, a vertical shaft is sometimes sunk, up which the minerals can be raised more quickly and cheaply than up an inclined shaft. It is also convenient for raising and lowering the miners.

Where the mineral does not lie in a vein, as shown in the last figure, but in great masses or lumps, it is reached by vertical shafts, or inclined shafts, levels, or open quarryings, according to the circumstances of the case.

The unwatering of a mineral field is frequently the most expensive preliminary to working the minerals; and, as above mentioned, water-levels have been driven upwards of 12 miles in length in some parts of Europe.[1] On the occasion of a recent

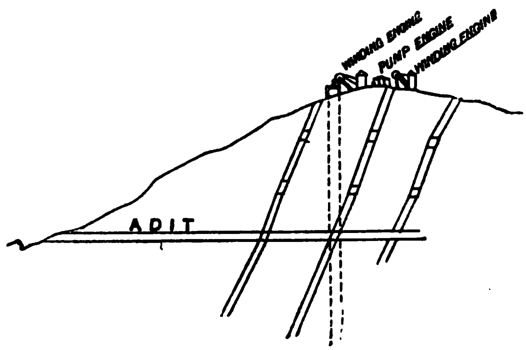


FIG. 125.—Winning metalliferous veins by inclined shafts, vertical shaft, and water-level.

visit to the Forest of Dean, the writer was struck by the great economy that might be effected in the working of the coal and iron mines in that district, by driving a level 2½ miles in length from the estuary of the Severn to the coal-field; this level might afterwards be continued across the coal-field for a total distance

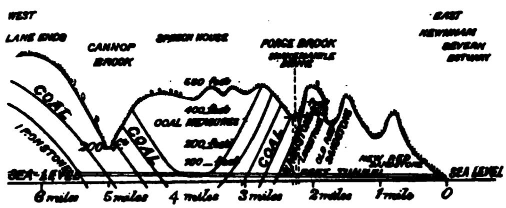


FIG. 125*a*.—Forest of Dean coal and ironstone field, with proposed water-level.

of 6 miles. By this means the height to which the water would have to be pumped would be reduced at the largest pumping-station by 360 feet, and in the other pumping-stations by heights varying from 200 up to 500 feet. Not only might this economy be effected, but if the rights of the mill-owners on the rivers were

[1] Schemnitz, Mansfeld, Clausthal.

bought up, water might be taken down the shafts to work hydraulic machinery, as the new tunnel would form an efficient tail-race. The mines in this district are let by the Crown, which would, therefore, at first sight appear to be the most suitable authority for undertaking this important work. A section illustrating this suggestion is given in Fig. 125*a*.

CHAPTER V.

SINKING AND SINKING MACHINES, TUBBING, WALLING, QUICKSAND, PNEUMATIC, FREEZING, KIND-CHAUDRON, ETC.

Site of Colliery.—In choosing the site of a colliery there are two sets of considerations—those referring to the surface and those referring to the minerals. Taking the surface first, the pit would be conveniently placed near a railway, or canal, or high-road, by which means of conveyance the produce could be despatched, and machinery and materials brought to the mine; on the other hand, a colliery might be most conveniently situated on that side of the estate where the minerals are deepest, because with a pit sunk at that point the pumping-engine in that pit would drain the whole estate of water, and it would be unnecessary to employ hauling-engines to drag the coal to the pit-bottom, as the coal could be brought to the shaft either by gravitation or by horses. It might, however, happen that the place suggested by mineral considerations might be on the top of a high mountain, or separated from the railway by a broad river, or by some strip of land over which there was no right of way, or some other impediment might exist which would overrule the mining considerations. Supposing, as far as the surface is concerned, it is a matter of comparative indifference where the shafts are sunk, then, in the case of a large estate, it is generally best to sink the pits as near the centre as possible; as there will be sufficient coal to the rise to last for twenty or thirty years. The pits will not be so deep as if they had been sunk at the extreme dip of the estate, and the length of haulage—that is to say, the distance that each ton of coal will have to travel underground from the working place to the shaft—will be on the average little more than half what it would be if the shafts were sunk at one side of the estate; and when it becomes necessary to get the coal from the dip side of the estate, it can be drawn by hauling-engines to the pit-bottom. In most deep collieries the amount of water finding its way into the dips is not a very serious consideration. For a large colliery it is necessary to have a railway, and it is an advantage to have also a canal and a good road. Pits are sometimes most advantageously situated on the side of a harbour or deep water inlet of the sea, so that the coal can be tipped directly into ships.

Sinking.—The site of a colliery having been chosen, the position of each pit is marked out. There must always be two shafts, and there are often three—one for pumping the water, one for a downcast shaft for the air, and the third for an upcast shaft for the air. In deep collieries the pumping-engine shaft seldom goes to the bottom, as the lower strata are dry; it is necessary that between two of the shafts, at each of which there must be a winding-machine, there should be a thickness of not less than 15 yards of strata; this is fixed by the Mines Regulation Act, 1887, sect. 16, sub-sect. B.

Unsoiling.—The first operation of sinking is to take off the soil for a depth of say 1 foot over the area likely to be occupied by the works and spoil-heaps, removing it to some place where it will be out of the way; this is a good practice, but not always observed nowadays, owing to the expense. The reason for it is that, after the mine is abandoned, the soil can be again spread over the ground, which will thus be restored to agricultural condition. In the case of collieries, however, likely to last a great many years, it probably does not pay to incur the expense of removing the soil for this purpose.

Excavation, and Shape.—The second operation is to begin digging the pit, or shaft, with picks and shovels. In England the pit is usually circular; in South Wales some pits have been sunk of an oval shape; and in Scotland there are many rectangular pits. On the Continent pits vary in shape, and in the United States the pits are generally rectangular. The size of colliery shafts in England nowadays varies from about 6 feet to 21 feet, and the usual size for first-class collieries, that is, collieries raising 1000 tons a day and upwards, is from 14 to 18 feet in diameter. By means of the pick and shovel alone, the depth of the pit will not be taken more than 3 or 4 yards, the dirt being thrown up on to intermediate platforms; it is then necessary to employ windlasses to raise the material in buckets, by which means the pits can be carried down to a depth of 20 yards. Twenty-five years ago and more it was usual to get a horse-gin, with which the depth of the pit could be increased to 40 or 50 yards, and after that a steam-engine was employed; nowadays a horse-gin is seldom used for colliery work, and a steam-engine is erected at once. As soon as the solid strata are reached, the pick by itself will make but slow progress, and it is necessary to employ gunpowder or other explosive.

Blasting.—To prepare holes for blasting, a drill may be used 6 to 10 feet long, made of iron or steel; if the former, with a steel end. The bit will be 3 inches wide to drill a hole 3 inches in diameter, if gunpowder is to be used. If dynamite is used, a bit 1½ inch wide will be sufficient, although 2-inch holes are sometimes used for dynamite. One or two men will take hold of the drill (see Fig. 126), lifting it up and down and striking blows at the bottom, turning the drill between each stroke, pouring water down the hole to assist the action of the chisel, and cleaning the hole of sludge with a scraper. For hard ground a shorter drill is used, which is struck by a hammer, one man holding the drill and turning it whilst the other strikes, and sometimes two men strike. "Sumpers" are generally drilled at an inclination of from 15° to 35° from the vertical, and side shots perpendicular; single shots are seldom drilled much deeper than 3 feet in hard ground, though in soft ground they may reach 4 or 5 feet. The miner sets a shot say in the centre of the pit, and the blast breaks up the rock around the hole, which, when cleared away, forms an excavation. Shots are now set round this first excavation to break off the sides until the whole width of the pit is extracted. Great judgment must be displayed in selecting the place for the holes, and the direction and depth of the boring. Frequently three or four shots are drilled and charged together, the fuse of each being lighted at the same time, but they are so arranged as not to go off simultaneously, owing to the difference in length of the fuse from each hole; so that the separate detonations of the shots can be counted, and the miners know if any have missed fire. Sometimes a number of shots are exploded simultaneously by means of electrical blasting. Fig. 127 shows the plan and section of a pit. Six holes are shown in and near the centre; these are the "sumping" shots, and are drilled to a depth of 6 feet; they are each charged with 2 lbs. of dynamite, and their simultaneous discharge blows up the whole centre of the pit, as indicated in the section. When the dirt from this discharge has been cleared away, ten more holes are drilled round the sides, as shown in the plan; and they are each charged with 2 lbs. of dynamite and discharged simultaneously, breaking off the sides, and filling up the pit with *débris* as indicated in the sketch.

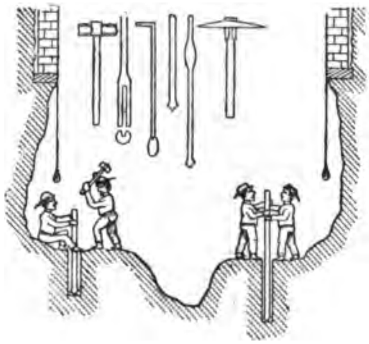


FIG. 126.—Sinking pit bottom, showing methods of drilling and tools used.

The ordinary method of charging a blast-hole is shown in Fig. 128. If gunpowder is used, the hole must either be made dry by putting in dust, which, when cleaned out, takes away the

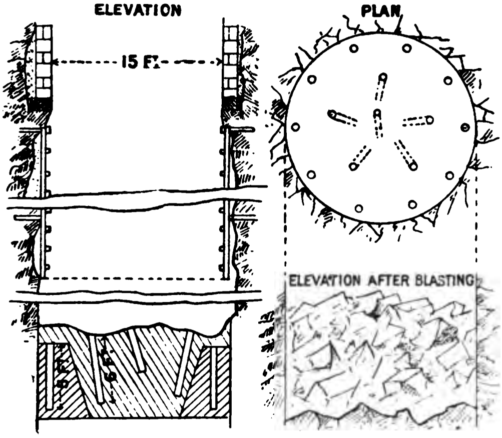


FIG. 127.—Simultaneous blasting by electricity. Sump shots, 6 feet deep; side shots, 5 feet deep; 2 lbs. dynamite each shot.

water, or else the powder must be in a waterproof cartridge-case —such a case is sometimes made of paper and sometimes of calico. The end of the fuse is put in the middle of the powder, and then the neck of the bag or case is tied fast round the fuse. Melted pitch may then be placed all over the bag and over the junction with the fuse; when this is dry it makes a good water-proof cartridge. The cartridge with the requisite length of fuse attached, say 4 or 5 feet, is lowered to the bottom of the hole, against which it is pressed with a scraper. Pounded shale or clay, dry and not gritty, is then pushed into the hole, and gently pressed, and very gently rammed upon the cartridge; more stuff is pushed in and rammed rather harder; small bits are put in and rammed. When about 8 inches of ramming is put in, the ramming is still harder. The rammer is generally made of iron, with a copper or brass end, or else it is entirely of brass. The hole is tightly rammed up to the top. When all the holes it is intended to fire, say three, four, or five, are rammed, the fuse is bent across a candle, which is not lighted. The tools are all sent out of the pit, and all the workmen go

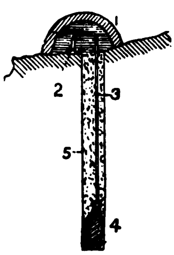


FIG. 128. — Shot-hole. 1, clay cover to protect flame; 2, candle, in clay, firing fuse; 3, fuse; 4, powder in tarred bag; 5, tamping.

out but one or two. A signal is given to the engine-man when a blast is about to be lighted; the short pieces of candle under each fuse are then lighted; the man jumps into the hoppet and signals the engine-man, and is immediately drawn up. The candle burns through the exterior covering of the fuse until it lights the powder in the fuse. As soon as this is lighted a fizzing is heard, and the fuse is said to have "struck." The miner is landed at the top, and the bridge drawn over, and the number of explosions counted, so that the sinkers may know that every shot has exploded before they descend. If the rock is hard, fragments may be thrown to a great height. If dynamite is used, a waterproof cartridge-case is not necessary, as a short immersion in water will not injure the dynamite; it is also not absolutely necessary to put in any ramming, or "tamping," as this ramming is called, because, owing to the great velocity of the explosion, water in the hole constitutes sufficient tamping; but, as a rule, tamping is employed, better results being obtained, although it is not necessary to ram so hard. The same kind of fuse is employed as for gunpowder. The dynamite does not explode with a simple flame or spark; it is necessary to employ an explosive cap or detonator to fire it. This is a copper cap containing fulminate of mercury; the fuse is slipped into the cap, which is squeezed tight with a pair of nippers, and some grease is put over the end of the cap to prevent water getting in, which would destroy it; the cap is pushed into the middle of a piece of dynamite. Dynamite is about three times as powerful as gunpowder, so that 2 lbs. of dynamite will do about as much work as 6 lbs. of powder. Gelignite or blasting gelatine are now generally preferred to dynamite.

Electric Blasting.—A method of electric blasting will be understood on reference to Fig. 129. The electric fuse A consists of two insulated copper wires, the bare ends of which are brought close together, and between them is placed an explosive composition called the priming, that is fired by the passage of the electric current; this composition ignites the small capsule of gunpowder, the flame from which ignites the main charge. If dynamite is used instead of gunpowder, the exploder is a copper cap, B (Fig. 129), containing an electric fuse similar to the above, but instead of gunpowder in the cap there is a charge of fulminate of mercury, the explosion of which makes the required flame and detonation combined, which causes the explosion of the dynamite. The copper wires (insulated) of the fuse are generally 4 or 5 feet long, or more for a deep blast-hole. If only one shot is discharged, each of the fuse wires is attached to a conductor coming from the battery. (This is on the pit-bank, and the conductors are taken down the shaft-side in waterproof covering.) If several shots are to be discharged at once, the right-hand wire of one fuse is connected to the left-hand wire of the next fuse (see Fig. 129). The left-hand wire of the last fuse in one direction is connected to one conducting cable from the battery, and the right-hand

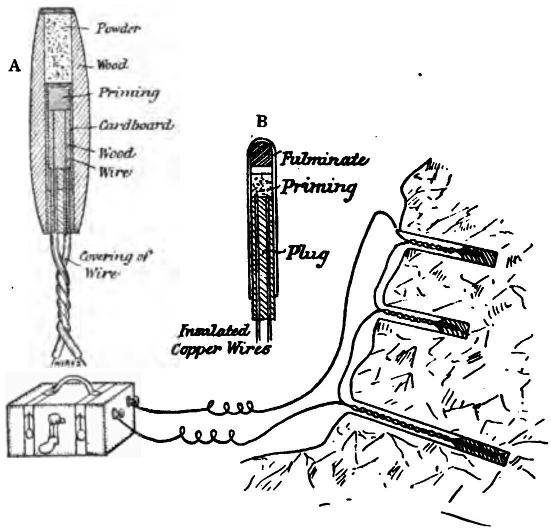


FIG. 129.—High-tension blasting. A, section of detonating-cap for dynamite, half real size.

wire from the last fuse in the other direction is connected with the other conducting cable. This is called connecting "in series;" the electric current has to pass through all the fuses in one circuit. The conducting cable is generally a strand of twisted copper wire covered with some insulating material, which is protected with plaited hemp or with tape, often with both. Another mode of discharging (see Fig. 130) is to connect the right-hand wire of each fuse directly with the right-hand cable by means of a short connecting piece of wire, and to connect the left-hand wire of each fuse with the left-hand conducting cable. The current in this latter case, instead of passing in circuit through the fuses, is divided amongst them all. This is called connecting "in

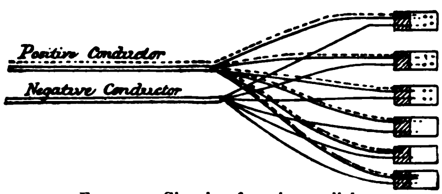


FIG. 130.—Showing fuses in parallel.

parallel." A magneto exploder is often used for high-tension fuses—the kind hitherto most generally adopted. In this machine, by turning a handle, and by means of multiplying gear, an armature is made to revolve rapidly between the two poles of a magnet, thus producing an electric current which passes in circuit through the machine; by pressing a button this circuit is broken, and another circuit, through the conducting wires and fuses, is completed; this heats the priming, and so causes an explosion.[1]

With a properly arranged and sufficiently powerful magnetic battery, as many as a dozen shots may be discharged at once. With small batteries, such as are usually supplied, five or six shots only can be discharged with certainty.

Lately, low-tension fuses have been adopted, and are recommended by high authorities (see Fig. 131). The external appearance of the fuse is much the same, but instead of the priming

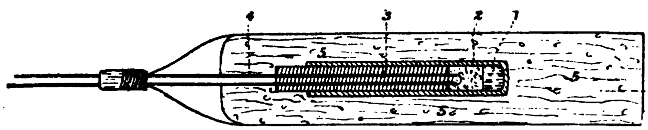


FIG. 131.—Low-tension incandescent detonator in cartridge. 1, fulminate; 2, platinum bridge in priming composition; 3, plug; 4, wires connected by bridge; 5, dynamite.

being fired by a spark passing between the two conductors, they are joined by a bridge of platinum wire, and the passage of the current causes this wire to be raised to incandescence, and so fires the priming. The low-tension system of blasting possesses the advantage that each fuse can be tested before being taken down the pit. This is accomplished by passing a very slight current through the fuse by means of a galvanometer. If the needle of the galvanometer moves, the fuse is good, but if it remains stationary the fuse is defective. The electric current may be produced either by a magneto exploder or by chemical action, the electric generator being a galvanic battery. A small one suitable for shot-firing in coal weighs about 2½ lbs., including a strong wooden case.

Messrs. Bickford, Smith, and Co. make a fuse for simultaneous blasting; for this purpose a slow-combustion fuse, burning at the rate of say 3 feet a minute, is connected to a tin tube in which is a small explosive charge. Into this tube are fixed the fuses

[1] Leyden jars charged by friction machines may also be used for shot-firing. The high tension of this current enables it to be passed great distances through thin wires.

connected with each shot; these fuses are made to burn at a very high speed; the explosive charge in the tube ignites all these high-speed fuses at the same instant, and as owing to the high speed of combustion in the shot fuses a short difference in length has no appreciable effect on the time, there is practically an immediate and simultaneous discharge.

Shaft-Lining.—The shaft having been sunk a few yards in depth, it becomes necessary to protect the sinkers against injury from stones falling down the sides. By careful inspection and picking off loose stones, the pit may be carried to a depth of say 8 yards. It is now lined with a wall of brick or stone; a bricking crib or curb of oak or cast iron (Fig. 132) is laid at the bottom and bolted together, the ground having been first cut perfectly smooth and level all round, leaving a deep hole, or sump, in the middle. This curb having been laid, the masons begin building on it. The walling is generally 9 inches thick, but the topmost length next to the surface is frequently 18 inches or 2 feet in thickness. By means of a plumb-rule the masons build the wall perfectly straight, any cavities behind the wall being filled up with small rubbish; in soft ground near the surface the wall is sometimes backed with concrete. In proceeding to sink below this first length, the ground directly under the curb is not cut away for a depth of between 2 and 3 feet below (Fig. 126), but after that the shaft is belled out to its original diameter, and the sinking continues for say another length of 8 yards. In order that the sinking may be plumb, plumb-lines are hung from the first curb with a weight at the end; these lines, suspended at intervals of about 6 feet all around the shaft, form a good guide for the sinker, and also for fixing the next brick curb. As an additional security that the second curb is fixed exactly under the first, a centre-line is hung down the shaft; this centre-line is hung from a strong piece of timber fixed securely across the centre of the shaft, either permanently or, if that is inconvenient, temporarily in permanent slots provided for it. A heavy plumb-bob at the end of the centre-line holds the line in the centre of the pit; by means of a rod the

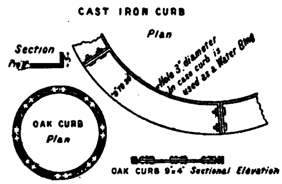


FIG. 132.—Walling curbs.

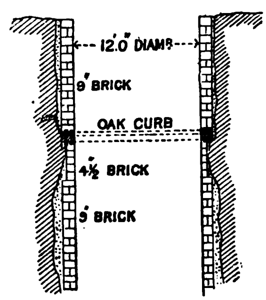


FIG. 133.—Section of shaft, showing brick walling.

distance of the curb from this line can be measured. Centre-lines are now often suspended from a movable jib; a very good form is made and patented by Mr. Wm. Foulstone, of Barnsley. The masons build up the wall on the curb, as before; when the wall is built up to within 3 feet of the wall above, some of the shale underneath the curb is cut out for a width of 4½ inches all round, leaving 4½ inches of solid ground to support the curb above. Brickwork 4½ inches thick is now built up to the curb above (see Fig. 133). In some large shafts the brickwork is thicker than 9 inches, being 14 or 18 inches thick (see Fig. 134). When the masons are building the wall up, and it has risen to a height of 4 feet, it is necessary that they should stand on a platform; this platform is sometimes made by fixing cross-bearers in holes in the brickwork, upon which are laid planks, the masons thus building up their own scaffolding as they proceed.

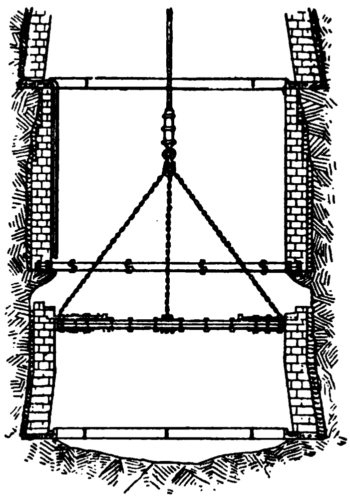


FIG. 134.—Operation of walling and suspended scaffold.

Scaffold.—A more expeditious and convenient plan, however, is to suspend a circular platform in the pit by means of a strong rope from the capstan at the surface. On this platform the masons stand, and upon it are laid the bricks and mortar. As the wall rises the platform is also raised (Figs. 171 and 134). In order to steady the platform, wedges are driven in at each side; sometimes bolts fastened through staples in the platform are knocked into holes in the wall, and this prevents the platform upsetting in case of uneven loading. The platform is frequently made with hinges in the middle, so that it can be folded up into a half-"moon" and withdrawn from the shaft or suspended from a hook in one side when it is not in use.

Capstans.—Fig. 135 shows a drawing of a capstan of the old-fashioned kind, with a vertical drum and six long wooden arms. This drum is usually worked by men—as many as twenty men could push at these arms at once; sometimes horses are used as well as men. Fig. 136 shows a crab capstan; in this case the drum is horizontal, and spur gearing is used to give the men at the handles a leverage. Instead of men, steam-engines are generally used, as shown in Fig. 136, in which a couple of small steam-cylinders, operating through three sets of spur-wheels, give movement to the drum. The chief dimensions of such a machine are as follows: pair of steam-cylinders, 10 inches in diameter, 18 inches stroke; horizontal rope-drum, 4 feet in diameter and 5 feet wide; pinion on fly-wheel shaft, 15 inches in diameter, gearing into wheel on intermediate shaft 6 feet 3 inches in diameter; pinion on intermediate shaft, 15 inches in diameter, gearing into wheel on drum-shaft 6 feet 3 inches in diameter; steel rope on drum of patent crucible steel, 5½ inches in circumference, galvanized, to lift say 10 tons. Such a capstan is stronger than required for a bricking platform; it is suitable for pumps and heavy machinery.

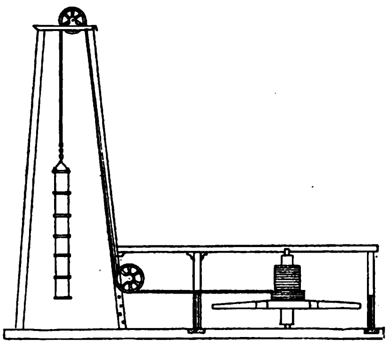


FIG. 135.—Old-fashioned capstan and shear legs.

This steam-capstan is much superior to either of the other two; it is much quicker than any hand-crab, and much safer than the old-fashioned capstan, and in modern works is nearly always erected in preference to the old class of machinery. The capstan ropes were formerly made of hemp about from 3 to 4 inches in diameter; they are now generally made of steel and galvanized to prevent them from rusting. The capstan rope, as shown in Fig. 135, goes underneath a pulley in the frame, which is fixed at about the level of the drum, and then goes straight up and over a pulley at the top of the frame, whence it descends into the pit. There should be a strong beam between the capstan and the pulley-frame, otherwise, when there is a heavy load in the shaft, there would be a tendency to pull the leg of the frame carrying the lower pulley towards the capstan, and this might lead to a serious catastrophe.

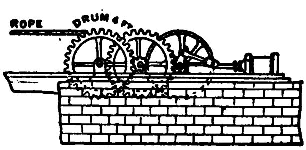


FIG. 136.—Steam capstan.

Pit-Top, and Winding-Engine.—Fig. 137 shows in side and end elevation the general arrangement at the pit-top. It is necessary that the mouth of the shaft should always be protected, and this is accomplished by a rolling bridge, to the end of which a fence is attached.

The banking arrangement is shortly as follows: the full hoppet having been raised well above the shaft-mouth, the bridge (*a*) is rolled across. On this bridge are rails which enables the waggon (*b*) to be run off and on; the contents of the hoppet are then emptied on to the waggon, the bridge rolled back, and the

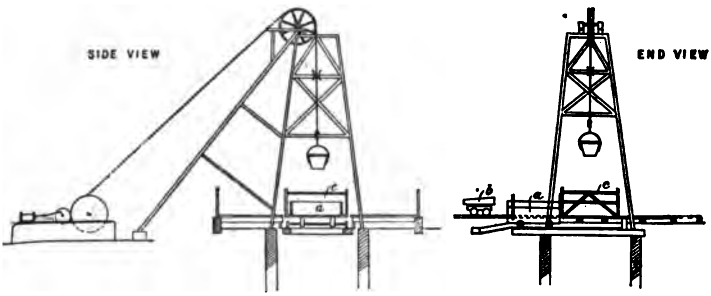


FIG. 137.—Sinking pit-bank.

waggon run on to the tip and emptied. Fig. 138 shows a different arrangement. Fig. 139 shows a suitable pair of engines for winding the dirt out of the sinking pit. The larger the engines are the better, but small engines are generally employed, in order to avoid the delay consequent on the erection of large engines. A pair of engines with cylinders 18 inches in diameter, with a stroke of 3 feet and a drum 6 feet in diameter, the leverage of the engine being multiplied by strong spur gearing in the ratio of 1 to 2, is a suitable size for sinking a pit for depths not exceeding 500 yards. A single rope only is used. At the end of this rope is a piece of heavy chain called a bull-chain, 3 or 4 yards in length, and at the end of that is attached the sinking-tub or hoppet. The heavy chain is required because, when the hoppet is detached, the weight of the rope on the other side of the pulley would overbalance the short length at the pit-side

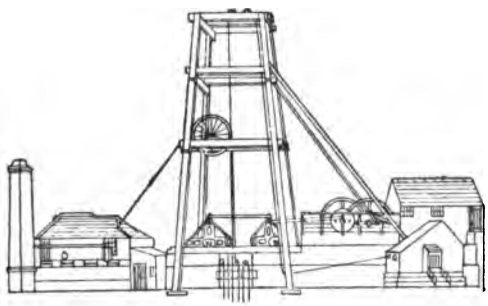


FIG. 138.—Pit-bank buildings for pumping and winding engines and pit frame.

and draw the end back over the pulley; this is prevented by the weight of the bull-chain. Formerly a flat rope was used for sinking, because round ropes were so liable to twist round with the hoppet, and would spin round in the shaft like a teetotum; latterly round ropes have been manufactured which do not twist, and as round ropes are cheaper and lighter than flat ropes, they are therefore preferred.[1]

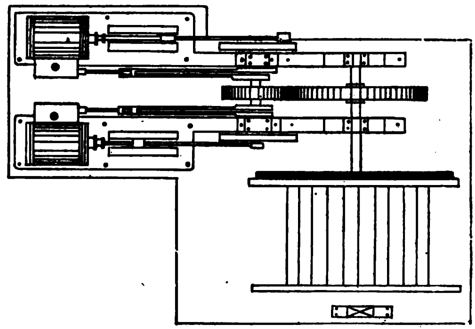
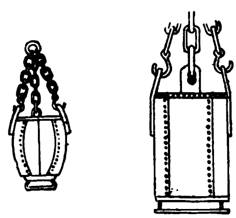


FIG. 139.—Pair of sinking engines.

Hoppets.—The hoppets used are as follows: Fig. 140 is an old-fashioned kind, and it is shown to be wider in the middle than at the top and bottom, like a barrel, and is attached by three chains to a ring at the end of the bull-chain; when the hoppet has to be detached, the three chains are each unhooked from the lugs on the hoppet. This barrel-shaped hoppet has the following advantages: When it is loaded with stone, the stone does not easily tumble out, being jammed in, owing to the



FIGS. 140, 141.—Sinking-tubs.

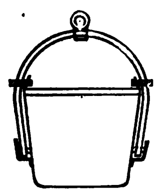


FIG. 142.—Water-barrel.

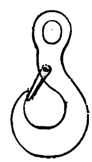


FIG. 143.—Hook for water-barrel.

upper parts being of less diameter than the middle. If when being wound up it should come in contact with the sides, it is the widest part that is most likely to touch, and there is nothing there to catch. Fig. 141 shows a hoppet with straight sides; this is also larger and holds a greater weight, and is more easily emptied by the banksmen at the surface. Fig. 142 shows a water-barrel; this is suspended from a snap-hook (Fig. 143) at the end of the bull-chain; the hook is fixed in a ring attached with a swivel-joint to a strong

[1] "Locked-coil" rope, and others.

iron bow, which is fastened by two trunnions to the water barrel just below the middle of its height, so that the barrel, when filled with water, can be very easily upset. To prevent it tipping up there is an upright pin on one side of the top edge, a collar sliding on the bow fits also over this pin; when the barrel is drawn to the surface, the banksman lifts the collar and thus tips the barrel. When dirt is being sent out of the pit, the hoppet is lowered down to the bottom, the rope is unhooked and sent up to the top, and a second hoppet sent down. As soon as the first hoppet is filled, it is sent up, and the sinkers proceed to fill the second hoppet. As it may take some time to empty the hoppet on the bank, especially if the dirt has to be taken some distance away from the pit-top, five or six hoppets may be necessary for each winding-engine. Hoppets of the form of Fig. 142 are now generally used for raising dirt, there being two collars, one on each side of the bow; the hoppet is wider at the top than at the bottom, and is large enough to hold from 1 ton to 3 tons of stone. It is emptied into a tip-waggon without being disconnected from the rope. In cases of hoppets used for water-winding only, a large valve may be put in the bottom so that it may fill itself if lowered into the water.

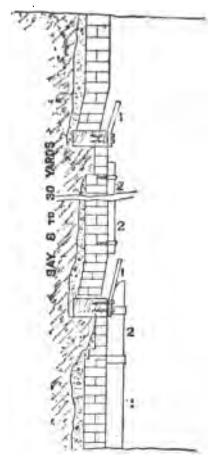


FIG. 144.—Section of shaft wall. 1, water-garlands; 2, water-pipes.

Water-Garlands.—In sinking a pit through ordinary ground, water is met with, and the amount of this water soon becomes sufficient to cause a good deal of trouble; in order to prevent it splashing about unnecessarily and hindering the men it is collected. For this purpose garlands are fastened on to the bricking-curbs (see Fig. 144). These are generally made of sheet iron about 12 inches wide, nailed all round the shaft, so as to make a ring standing two or three inches out from the brickwork; all the water trickling out of the brickwork must run into this ring; at one place a hole is made in the ring out of which the water can run into a pipe, say 2 or 3 inches square inside, sufficient to carry the water. At the curb below is another garland. The pipe delivers its water into this second garland, which also collects fresh springs; a second pipe goes down from the second garland, which may be larger in size than the first one, because it may have to take a larger quantity of water. In this way the water is conducted to the pit-bottom or to some cistern in the shaft-side. If it goes to the pit-bottom, the wooden pipe stops at the lower end of the brickwork, and a flexible tube (or simple rope) continues it to the bottom, where it delivers the water into the water-barrel. As soon as this barrel is filled, it is wound up by the engine, and the flexible water-pipe is put into another barrel which is ready at the pit-bottom. In this way small feeders of water can be raised. If there is more than a small supply, such as 50 gallons a minute, a separate winding-engine is necessary to wind the water; and if the quantity of water is more than 200 gallons a minute, pumping-engines are usually employed. Cast-iron water-rings are often used. A good design is made by Needham Bros., of Barnsley, shown in Fig. 144*a*.

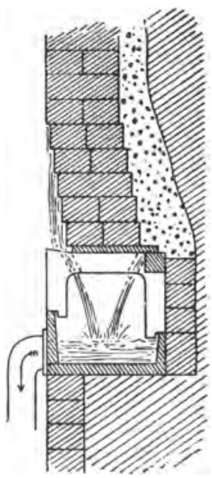


FIG. 144*a*.—Cast-iron water-ring.

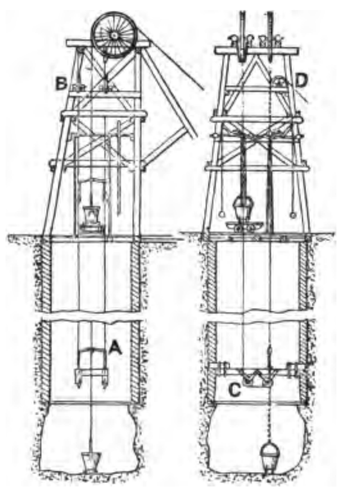


FIG. 145.—Galloway's guides.

Guides.—As a general rule, the sinking hoppet is not attached to guides. After it has been lifted off the bottom, one of the sinkers steadies it, and when it has stopped swinging, it is wound up, and it swings but little on its way.

Some engineers, however, prefer to have the sinking hoppet guided. Amongst such methods one designed and patented by Mr. Galloway has been used in South Wales and elsewhere. It is shown in Fig. 145. This system of guiding may be described as follows: Two wire guide-ropes are stretched from the pit-frame on the top to a platform fixed at the bottom of the brickwork in the shaft (see Fig. 145). The cross-bar A, with a collar at each end, fits on to these guides; in the centre of the cross-bar is a hole through which the winding-rope passes; beneath the cross-bar is attached the hoppet. When the hoppet is raised, a rubber buffer at the lower end of the winding-rope comes in contact with the cross-bar, so that it also is lifted. As the cross-bar is connected with the wire guides, it cannot swing, and the hoppet is also steadied. In lowering the hoppet to the bottom, the cross-bar is arrested by some buffers on the guides just above the platform, but the hoppet continues to descend through an opening in the platform to the bottom of the pit. The wire guides are generally attached to capstans, by means of which they can be raised or lowered, and the platform is used for the masons in walling the shaft. In large pits there may be two hoppets and two sets of guides; in this case there is only one capstan rope for each set of guides (see Fig. 145). One end of each wire guide is fastened to

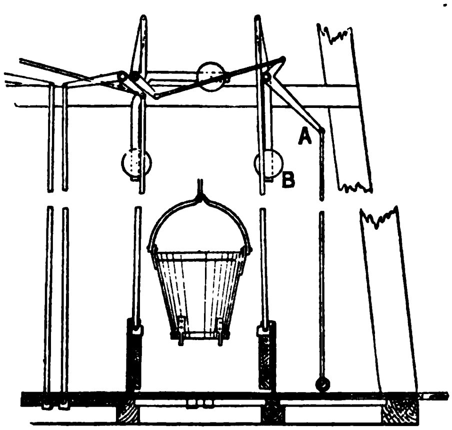


FIG. 146.—Galloway's guides: levers for opening doors.

the pit-frame at D; it passes down and under the pulleys C in the pit, then up to the top again over the pulleys B, and thence to the capstan.

The pit-top is covered over, and there is a pair of doors through which the hoppet can be raised (see Fig. 146). These doors are lifted by means of the lever A and the balance-weights B. When they are open they form a fence for the shaft, the other two sides of the opening having permanent fences. After the hoppet has been raised, the trap-doors are closed, and a bridge run over them, into which the hoppet is lowered.

Sinking-Pumps.—The pumps most commonly employed in sinking pits are shown in Fig. 138. On the crank-shaft of the steam engine is a small spur-wheel or pinion, which gears into another spur-wheel three or four times the diameter of the pinion. On the same shaft as the large wheel is a crank or disc, with three holes for a crank-pin, which can be placed in either hole to make a long or short stroke as required. A connecting-rod from the crank-pin gives movement to beams shaped like the letter T inverted, or the letter L, at the pit-top. The pump-rods are attached to these beams by the method shown in Fig. 147. The beam is made of two sides, a pin, A, passing through both; this pin also holds a saddle, B, which can turn upon the pin. By means of clamps C C, this saddle is fastened to the pump-rod. These pump-rods are of wood, and from 4 to 12 inches square, according to the size of the pumps, 6 inches square being a suitable size for pumps 14 inches in diameter working from a depth of 50 yards. The pump shown in Figs. 148 and 149 is that kind usually called a bucket-lift. The pipes are all made of cast iron. The bottom pipe has a rounded end perforated with holes, called a wind-bore; this rests on the ground, standing in the water; if the water should not be deep enough to cover all the holes, those above the water may be plugged, so as to prevent air from drawing in. Above the wind-bore is the clack-piece; the clack-valve is shown also in Fig. 148 (1). The shell, or valve-seating, is of cast iron, tapered on the outside to fit into a tapered recess in the clack-piece. There are two openings for the water, both of which are covered by a wrought-iron plate, which is the valve. This wrought-iron plate is riveted on to a piece of tough leather, which covers the whole surface of the shell, and is fastened down by a collar on the hook *a*, which passes through the shell; this leather forms the hinge, and also makes a soft bed for the valve, which would not otherwise be water-tight. A hook or bow is used for the purpose of lowering or raising the clack. The cast-iron door can be taken off when it is necessary to change the clack. It has sometimes happened that the door has been taken off, and could not be put on again before the water had risen up, so that the clack was useless. There is often another clack-seating above the door, where the clack could be placed in case the lower clack was rendered useless in any way. To avoid the risk of not having the clack-door on or properly tight, there is sometimes no

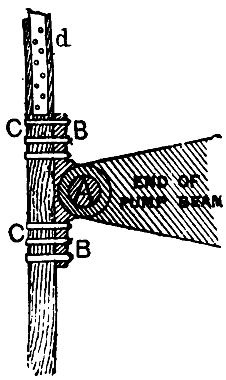


FIG. 147.- Pumping-beam and saddle.

clack-door, and the clack has to be drawn up to and lowered from the surface every time it is changed. Above the clack-piece comes the working barrel; this pipe is about 10 feet long, it is accurately bored, and is the cylinder in which the bucket works. Above the working barrel is a bucket-door piece, a short pipe 4 to 6 feet long, in which is a door similar to the clack-piece door. At this door the bucket can be taken out and replaced. This door is also sometimes omitted for the same reason as given above for dispensing with the clack-piece door. Above the bucket-door piece are the ordinary "pump-trees." The "pump-tree" is a pipe, so called because it was originally made by boring out the trunk of a tree. It is generally made of cast iron with flanges, and should be at least 1 inch in diameter larger than the working barrel, so that the bucket can be easily withdrawn or lowered, and the flanges should be faced in the lathe, so that the joint can be easily made water-tight with indiarubber rings. Bolts and nuts are used for fastening the joints. Formerly pump-trees were not faced in the lathe, and the joints were made water-tight by means of a thin wrought-iron ring, or washer, rather larger in internal diameter than the bore of the pump-tree. This ring was wrapped

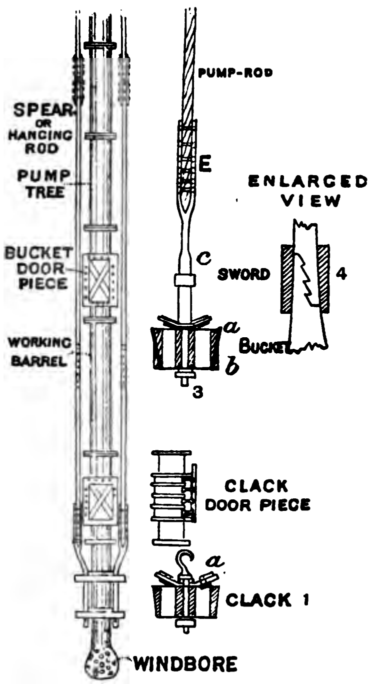


FIG. 148.—Pumping-set.

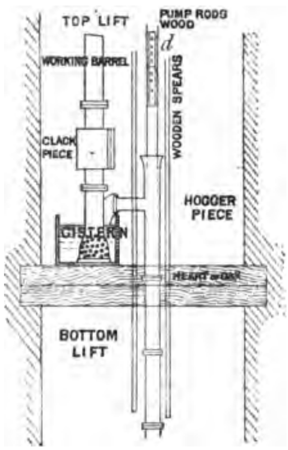


FIG. 149.—Pumping-set.

with flannel, and placed between the two flanges, and when the bolts were tightened the flannel served to make a water-tight joint. At the top of the pump-trees is the hogger-piece (see Fig. 149). This is a branch pipe for the delivery of the water, and is belled out at the top to prevent the water overflowing when the bucket is lifted. To the hogger-piece is attached a large flexible tube frequently made of leather, which conveys the water into the drain. Pipes of wrought iron and steel are sometimes used instead of cast iron. They have generally a riveted longitudinal seam, and the flange is riveted on by means of an angle iron, which is riveted round the pipe and also to the flange. These pump-trees of wrought iron or steel are very much lighter than cast iron, owing to the greater tensile strength and trustworthiness of wrought iron.

Bucket.—The bucket is shown in Fig. 148 (3). The valves are similar to those in the clack. The water is prevented from slipping between the bucket and the working barrel by a leather packing-ring, *a*. This ring is held tight by a tapered iron ring *b* driven on from below. When the bucket is lifted, the water on the top, pressing on the inside of the leather ring, forces it open and against the sides of the working barrel, and until this leather ring is worn out the water will not slip past the bucket to any large extent. The bucket is attached to the rods by a toothed iron or steel bar, called a sword, shown in Fig. 148 (3) *c* and (4), over which a ring is driven to keep the teeth in their places. The sword is attached to the pump-rods by the iron fork and by bolts, as shown in Fig. 148 (3) E. The rods, which may be in lengths of 20 or 30 feet, are joined one to the other by wrought-iron plates and bolts (Figs. 147 *d*, 149 *d*, and 150 *d d*). Whilst the working barrel is 10 feet long, the stroke of the bucket in a sinking pit seldom exceeds 6 feet, and is sometimes only 2 feet. As the pit gets deeper the pumps have to be lowered. When first fixed the bucket works to the bottom of the working barrel, but when the pumps have been lowered it works at the top. The bucket can be lowered by undoing the clamps above mentioned, and letting down the required length of rod, fresh rods being added at the top as required.

Pumps are frequently suspended by ropes, or chains, or wooden rods called spears; these latter are shown in Fig. 148. They are clamped to the wind-bore, and at intervals are attached to the pumps above by collars. The upper end of each is attached by clamps (Fig. 150) to a large screw. This screw-thread is from 6 to 10 feet in length (on a rod 6 feet longer than the screw)—the longer the better—and is from 3 to 4 inches in diameter, with a square thread chased. At the lower end of the rod is a head with an eye in it. To this eye is suspended by a pin the clamping-piece to which the rods are fastened. The screw passes through a nut, below which is a large washer; this washer rests on a strong beam or beams (Fig. 150). As the pit gets deeper the pumps are lowered by unscrewing these nuts. A long lever and ratchet is commonly used for this purpose. If the nuts were attached by gearing to a small steam-engine, the raising and lowering of the pumps would be more expeditiously and cheaply effected. When the pumps have been lowered the length of the screw, the clamps are loosened, the screws run up, and the clamps again tightened to a higher part of the spears. Instead of this system of screws, capstans and pulleys are sometimes employed (see Fig. 155). In other cases the pumps are not suspended, but are fixed to cross-bearers in the shaft, and the wind-bore slides over a pipe attached below the clack-piece, which has been turned in the lathe so as to make an air-tight joint with the sliding wind-bore. The joint is completed by means of a stuffing-box (Fig. 152). As the pit gets deeper the wind-bore can fall down the length of the slide, which may be 10 feet. In order to add fresh pumps, it is necessary to break the joint above the bucket-door piece, and lower that and the pipes below by means of the capstans sufficiently

Fig. 150.

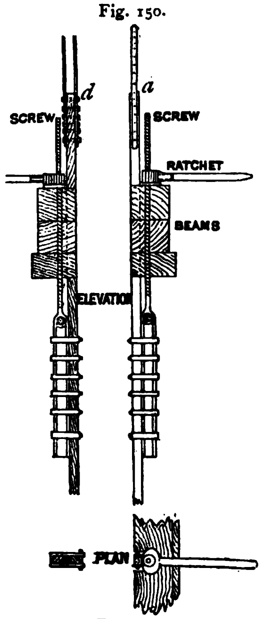


Fig. 151.

FIGS. 150, 151.—Pumping-set screws.

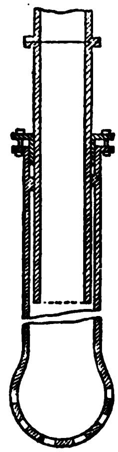


FIG. 152.—Sliding wind-bore.

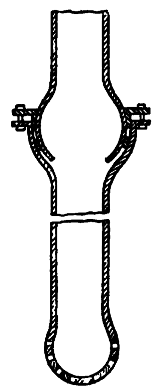


FIG. 153.—Swinging wind-bore.

to take in a 9-foot pipe, the slide being run up to the top of its length.

Swinging Wind-bore.—It is necessary to blast the ground immediately underneath the wind-bore. Where the pumps are suspended by spears, they can be swung from one side of the pit to the other, but where they are fixed, this cannot be done. To overcome this difficulty a swinging wind-bore (see Fig. 153) is sometimes used. This is made on the principle of a ball-and-socket joint, and it can be moved a distance of 7 or 8 feet in either direction. When blasting, the pumps are protected as

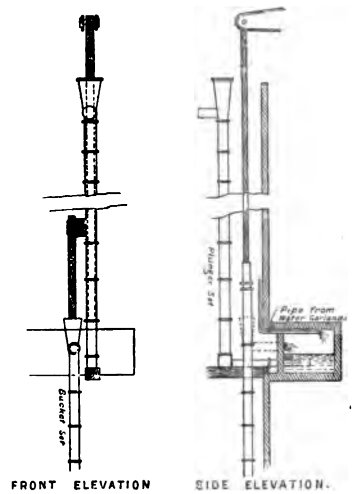


FIG. 154.—Forcing set raising water from chamber in shaft-side, supplied by bucket lift below.

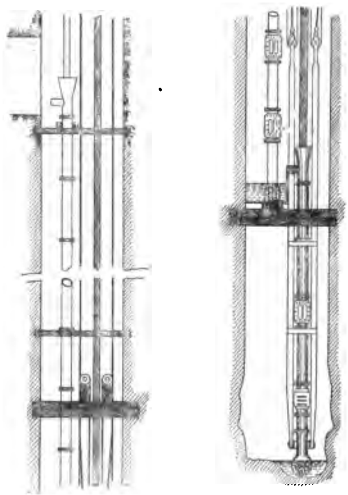


FIG. 155.—Pumps suspended by pulleys and ropes from surface.

much as possible by wooden planks from the effect of stones thrown across the pit.

It is seldom that an ordinary pump is employed to lift the water a greater height than 70 or 80 yards. When the pit exceeds that depth, another set of pumps is employed (see Fig. 149), which pumps the water from the bottom into a cistern, from which the upper set of pumps takes the water to the surface. Where there is a large quantity of water, several lifts have to be put in. As many as five lifts have been employed at one time working from the bottom, and these pumps are sometimes 20 and even 30 inches in diameter. When dealing with large quantities of water, the method of pumping may be modified by the consideration that the pumping-engines will be only temporarily required during the sinking, and that the water will afterwards be tubbed back, and the pumping-engines become useless. If, on the other hand, the pumping-engines will be permanently required, machinery of a permanent and economical kind may be erected. The kind of pumping-engine known as the Cornish has a great reputation for economy, and is therefore employed sometimes as a sinking-pump (see Figs. 154 and 155).

Where the pump is only temporary, it has of late years become common to suspend in the shaft steam-pumps by means of wire ropes (Fig. 156). The steam is taken down the pit in wrought-iron pipes with a sliding joint, so that the engine can be lowered as required. The delivery-pipe of the pump is free to be lowered down. The suction-pipe is sometimes a flexible tube or has a sliding joint, so that it can be kept at the bottom of the pit, or the engine is constantly lowered as the pit is deepened. These pumps are convenient, because, owing to the steam-cylinder and pumping-cylinder being fastened together on one iron plate, they require no external support, and they may be worked at a great pace with comparatively little risk of fracture: one of these pumps may deliver as much as 1000 gallons a minute. Three sets of pipes have to be suspended in the shaft—steam, exhaust, and water. These may be attached to cross-bars, as shown in Fig. 157. The delivery-pipe is brought up, say, 10 feet above the bank, and also the exhaust-pipe (see Fig. 156).[1] As the pit is deepened, the pump is lowered by the capstan-rope by which it is suspended. The steam-pipe, having a slide at the pit-top, lengthens. The delivery and exhaust pipes, not being attached at the top, are free to follow the pump downwards. Fig. 158 shows another kind of pump, which may be suspended in a similar manner. There are a variety of pumps used for this purpose, and those shown in the

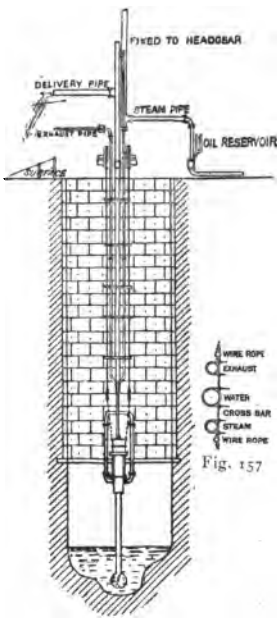


Fig. 156.

FIG. 156.—Suspended pump and arrangement of pipes in shaft.
FIG. 157.—Plan showing arrangement of pipes.

[1] See paper read at the Midland Institute, by Mr. W. H. Chambers, on the "Cadeby Sinking."

illustrations are not necessarily better than any of the others. The pulsometer (see Fig. 159) has also been used in shaft-sinking. As it is not suitable for high lifts, it is necessary to place one above the other; the bottom pulsometer delivering into a cistern from which the next one takes its water, and so on up to the surface. Fig. 160 shows a method that has been adopted in sinking the Canklow pits of Sir John Brown and Co.[1] At this pit the quantity of water met with near the surface was very great, amounting to between 2000 and 2500 gallons a minute. Nine pulsometers were employed in three tiers, each pulsometer raising

FIG. 158.—Worthington sinking pump.

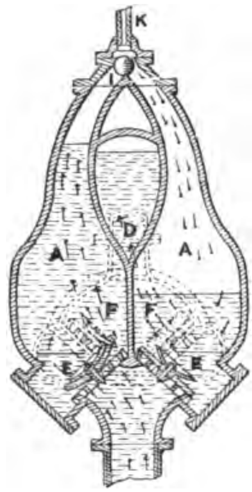


FIG. 159.—Section, showing pulsometer pump fitted with grid valves.

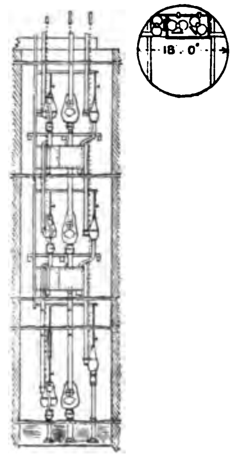


FIG. 160.—Pulsometer (Canklow pits).

the water to a height of about 72 feet. The steam was taken down the pit in pipes from boilers worked at a pressure of 80 lbs. per square inch, the sliding steam-pipe being used for the lowest set of pulsometers. The three lowest pulsometers were suspended by chains from three crab-winches on the surface; the two upper tiers of pulsometers were fixed on wooden beams. At this colliery, after the pit was completed, the water was kept out by tubbing, and the pulsometers were then no longer necessary. There is no way of ascertaining the amount of water delivered by a pulsometer, except by measuring the quantity delivered; but with regard to

[1] See *Engineering*, May 13, 1892.

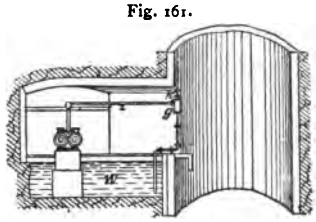
ordinary pumps the probable quantity of water may be calculated from the diameter of the working barrel, the length of the stroke, and the number of the strokes per minute.

Let D equal diameter in inches, y equal stroke in yards, and x equal delivery in gallons; then $x = \frac{D^2 \times y}{10}$. Then if $D = 10$ and $y = 2$, $\frac{D^2 \times y}{10} = 20$; that is to say, the pump will deliver 20 gallons per stroke, and, if there are ten strokes a minute, it will deliver 200 gallons a minute.

To take another example. Suppose $D = 20$, and the stroke is $1\frac{1}{2}$ yard, and the number of strokes per minute $= 10$; then $y = 10 \times 1\frac{1}{2} = 15$, and $x = \frac{20^2 \times 15}{10} = 600$ gallons per minute. If the pump does not deliver this quantity, it will be because there is some slip of water past the bucket, ram, or piston, or past the suction or delivery valve, or else because the water during the suction stroke does not follow up the bucket or ram to the end of the stroke, *i.e.* is not "pumping solid."

Underground Pumping-Engine.—It is often found convenient, when sinking a pit, to fix a permanent pumping-engine in a chamber by the side of the shaft (see Figs. 161 and 162 [1]), the steam being brought down to the engine in pipes. After the steam has done its work, it is sometimes condensed and then passed up with the delivery of water, or it is taken to the upcast shaft, or it is conveyed to the surface in separate pipes or behind a brattice; the advantage of this mode of pumping the water is that the steam and water pipes occupy less room in the shaft than the pumps worked by rods from the surface, and also they occupy no space on the surface. The heat from the steam-pipes and exhaust steam is useful for ventilating purposes, if they are in an upcast shaft or on the upcast side of the brattice in a shaft.

Fig. 161.



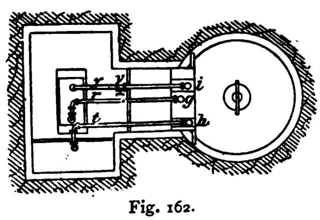


Fig. 162.

FIGS. 161, 162.—Sectional end elevation and plan of pump-chamber.

Where there is a great deal of water, the sinking of two shafts simultaneously facilitates the pumping, because there may be

[1] W. Galloway, Llanbradach.

pumps in both shafts; and if the water has to be tubbed out it is highly desirable that all the shafts at the colliery, whether two or three, should be sunk simultaneously, because, if the first shaft was sunk and tubbed, then all the water would have to be pumped at the second pit, whereas, when the pits are sunk together, one pumping does for both the shafts. If, however, the pumping will be permanent, it is sometimes economical to sink the pumping-shaft in advance of the other shafts; in this way the strata are drained, and the second and third pits may be sunk without any pumping. If, notwithstanding the existence of a pumping-shaft, there should still be water in the other shafts, they may be drained in the manner shown in Fig. 163. Here a heading is driven from the pumping-pit under the pits which have to be sunk, and then a bore-hole put down from the sinking pit into this heading, so that any water that is in the shaft runs down this bore-hole, and the sinkers are free from the hindrance of water. There may be some tendency for the hole to be stopped up. To avoid this a chain is sometimes passed through the hole and fastened above and below; if the chain gets jammed, it may be attached at the upper end to the engine or capstan, and pulled, and in this way the hole is cleared.

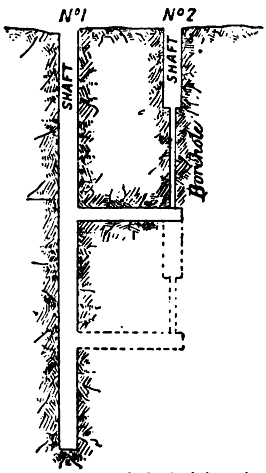
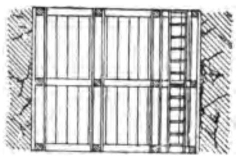


FIG. 163.—Bore-hole draining pit.

The foregoing remarks on sinking are taken from the experience of colliery shafts, but the sinking of shafts for other purposes, such as for iron, tin, or gold mines, is conducted on similar principles. It is, however, common, in the sinking of metalliferous mines, to sink shafts at an inclination from the vertical in the vein and following the vein, so that, whilst the shaft is straight as regards the direction by the compass, it may be crooked as regards its inclination. Where the shaft is inclined, the hoppet, instead of being suspended without guides, is frequently a box on wheels running on or between guides; in the same way the rods from the pumping-engine are supported by rollers. But where metalliferous shafts are sunk in the vein, the sinking is concurrent with the opening out of the mine, and therefore, after the first two or three levels are opened, the sinking of the shaft is gradual, like the development and exhaustion of the mine, and is continued for scores or even hundreds of years.

Rectangular Shafts.—Where timber is used for the permanent lining of the shaft, it is frequently made rectangular, as

Fig. 164.



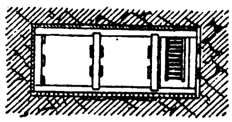


Fig. 165.

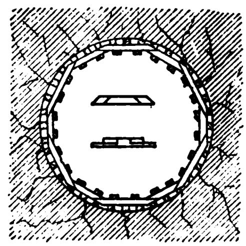


Fig. 166.

FIG. 164.—Rectangular shafts: section. FIG. 165.—Plan. FIG. 166.—Shaft nearly round.

shown in Figs. 164 and 165. There is a framework of round or square timber placed at intervals of from 3 to 6 feet, and behind this are planks fixed in their places by wedges; sometimes the lining is adapted to a nearly circular shaft (see Fig. 166), and sometimes to a shaft that is quite circular. Shafts lined with brickwork are sometimes oval in shape.

Temporary Lining.—In sinking a shaft it is often found convenient to put in a temporary lining for a length of 20 yards, and sometimes up to 50 or 60 yards, or even more, and then to put in the permanent lining of brickwork. The temporary lining is sometimes made of light wooden curbs of oak, say 4 inches wide and 3 inches deep, cut to a circle in segments and bolted together (see Fig. 167). This curb may be supported on iron pins placed in holes which are driven into the shaft-side and wedged tight, or it may be suspended from some fixed curb by wooden boards nailed from the curb above to the curb below; these boards, which are called lacing-boards, constitute the lining; the curbs are placed at intervals of 3 or 4 feet, and the lining-boards may be long enough to be attached to three or four curbs. Another method of lining used by Mr. Charles Walker in a sinking near Barnsley is shown in Figs. 168 and 169. In this case iron rings are used instead of

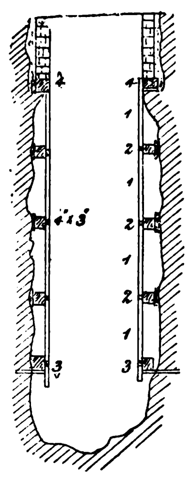


FIG. 167.—Temporary lining. 1, facing boards; 2, wood curbs wedged at back; 3, wood curbs resting on pins; 4, bricking curb.

wooden rings; the top ring is carried on iron pins wedged fast into holes driven into the shaft-side, the lower rings are suspended by iron hangers bent into the shape of a hook on the top and bottom. Boards are placed behind these rings and fixed tight by wedges. As the permanent brickwork is brought up, the lower rings are taken to pieces, and the lining-boards removed ready for use again.

Fig. 168.

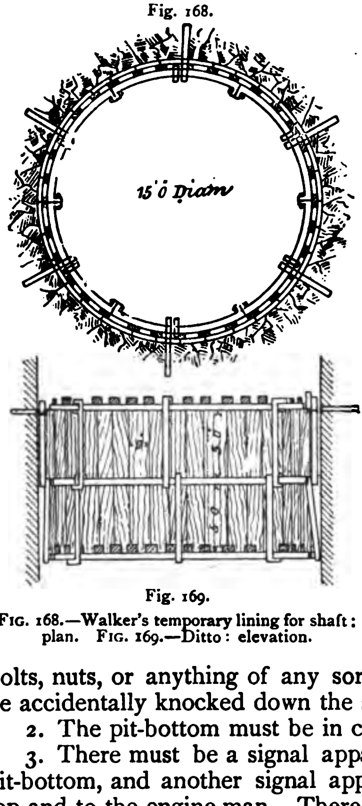


Fig. 169.

FIG. 168.—Walker's temporary lining for shaft: plan. FIG. 169.—Ditto: elevation.

Rules for Safety.—The only way to secure safety in a sinking pit is to employ experienced sinkers and to enforce a strict observance of rules. Some amongst the many rules that have to be regarded are as follows:—

1. The pit-top must be in charge of a banksman, whose directions must be obeyed by every one on the surface about the pit-top. The banksman must see that there are no loose materials, bricks, stones, dirt of any kind, loose picks, shovels, wedges, hammers, bolts, nuts, or anything of any sort near the pit-top which could be accidentally knocked down the shaft.

2. The pit-bottom must be in charge of a master sinker.

3. There must be a signal apparatus from the pit-top to the pit-bottom, and another signal apparatus from the bottom to the top and to the engine-man. There must also be a signal apparatus from the bank to the engine-man; it is also desirable that the engine-man should clearly see the pit-top.

4. There must be a good light on the pit-bank at night, but screens must be placed between these lights and the eyes of the engine-man.

5. When the sinking pit is entered for the first time after it has been left without any sinker in it, even for a brief period, the master sinker must go down with safety-lamps only, and examine for explosive gas, and particularly must this precaution be observed after blasting.

6. Before starting to work on the pit-bottom, the master sinker

and his deputies must examine the shaft-sides from top to bottom, and particularly those portions where there is no lining, and loose stones must be pulled off, so that no sinker may work beneath ground which is unsafe. The master sinker must particularly observe that every opening made into the shaft-side, such as a heading in a seam of coal, or for any other purpose whatever, is safely walled, so that no loose material can fall from the sides of the heading and fall down into the shaft. He must also see that every edge of wall, whether at the pit-top or at such openings, is securely fixed with timber or cement, so that no loose bricks can fall off. He must also see that there are no loose bricks, stones, tools, bolts, nuts, or any loose thing whatsoever upon any of the rings, ledges, bearers, pipes, or platforms in the pit.

7. A sinker must be appointed as hanger-on to give signals to the bank; he must also steady the hoppet whenever it has been lifted off the bottom, and must see that it has ceased to swing before he gives the signal to wind up.

8. The engine-man must not lower the hoppet to the bottom, but must stop at least 5 yards off the bottom until he receives the signal from the hanger-on to lower down. Unless this rule is observed, the sinkers might be crushed by the hoppet descending upon them.

9. The hoppet must not be filled over the top with dirt, and all tools reaching over the top must be tied firmly to the chains, so that they cannot fall out.

10. The pit-top must be quite covered over, either with trap-doors or with a movable bridge, whilst the hoppet is being emptied, or one unhooked and the other one being hooked on.

11. When the bridge is run over the pit-top it must be immediately and securely fastened, so that it cannot run back (else the banksman and the hoppet might be accidentally precipitated into the shaft).

12. There must be a code of signals for the guidance of the engine-man, which may be arranged to suit the previous experience of the workmen; as, for instance—

One rap to raise the hoppet off the bottom.

One rap to wind up after the hoppet is steadied.

One rap to raise the hoppet off the bridge.

Two raps to lower down.

Three raps to raise the hoppet off the bottom when men are about to ascend.

Four raps to go on slowly.

Six raps when a shot is about to be fired and to raise the hoppet just clear of the bottom, so that the sinker may get in after lighting the fuse.

One rap to go on.

13. Safety-lamps to be used at the pit-bottom when about to cut into a seam of coal.

14. A good ventilation to be maintained at the pit bottom, and especially both above and below the scaffolding when bricking.

15. Explosives to be taken down in cartridges and canisters.

A great many precautions not indicated in the above remarks have also to be taken. In some cases safety-hooks are used to safeguard against overwinding.

Ventilation.—For the ventilation of a sinking pit it is sometimes divided into two compartments by a vertical division,

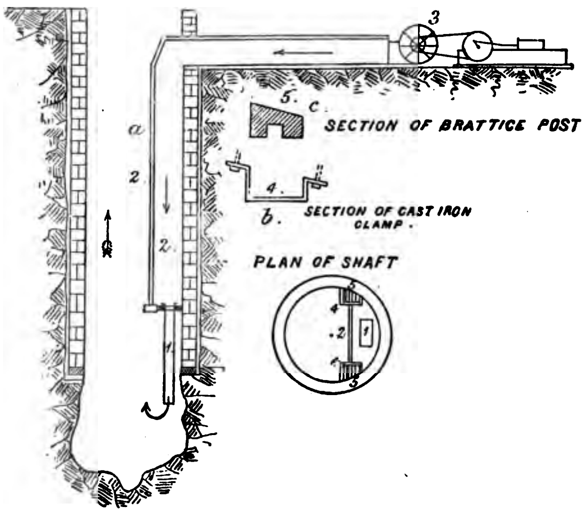


FIG. 170.—Bratticing of shaft. 1, air-pipe; 2, brattice boards; 3, fan; 4, iron clamp; 5, wood posts to hold boards.

called a brattice; such a brattice is shown in Fig. 170. Here it is made by fixing, with iron clamps (4) and coach-screws, two posts (5) to the shaft-side; in each of these posts is a groove wide enough to take a 2-inch plank. The planks are tongued and grooved, so that when fitted together and wedged into the groove of the posts, they make an air-tight division. Any space between the posts and the bricked lining may be stuffed with oakum or other material to prevent the air drawing through; whatever material is used it should be such that, if it should loosen and fall, it will not injure anybody by its weight. The bottom of the brattice is supported by a beam fixed into the shaft-sides; since the brattice cannot be continued right down to the bottom, a floor is put under to close it. At the level of the beam an opening is cut in the floor to which pipes are attached, say 15 inches by 20 inches internal dimensions; these pipes are carried down to within a few feet of the bottom. If there is a steam-engine behind the brattice, the heat from the steam-pipe will cause an upward current, or a steam-pipe may be taken to the bottom of the brattice and some steam allowed to escape, for the purpose of warming the air and causing an upward current; otherwise the top of the brattice must be connected with a fan driven by a steam-engine. A small engine, with a cylinder 6 inches in diameter and 12 inches stroke, would be sufficient for the ventilation of a single shaft, and for a fan 3 feet in diameter. The fan is generally an ordinary centrifugal fan. If the brattice is connected by means of a pipe with the centre of the fan, it will then suck air out of the shaft; but if the brattice is connected with the outside of the fan casing, it will then blow air down the brattice, because every centrifugal fan sucks in air at the centre and discharges it at the periphery. In some cases there is no brattice, but air-pipes are carried all the distance (see Fig. 171). If wooden pipes are used with an internal area of 2 square feet, they are sufficient for a single shaft under ordinary circumstances; but brattice facilitates the ventilation of the pit, and clears it of smoke. If pipes are used all the way, it is usual for the fan to blow air down them, because then the current will strike down from the bottom of the pipes and clear the shaft-bottom of smoke, after blasting, almost immediately, but the upper part of the shaft remains smoky. If, on the other hand, the fan sucks air, the whole shaft has to be cleared of smoke before the bottom is quite clear, and therefore, unless there is a powerful ventilation, there is a loss of time. When a bricking scaffold is in the shaft, the air-pipe must be continued past the scaffold, so as to ventilate the pit-bottom when the scaffold is raised up; but water may then accumulate at the bottom and close the end of the pipe, and this stops the ventilation. It is necessary, therefore, that there should be a branch pipe above the scaffold, so that in case the lower pipe is sealed with water, there may still be ventilation. This may be carried out by making the pipe that goes through the scaffold smaller than the main pipe, and leaving the bottom of the latter partially open, as shown in Fig. 171. Sometimes circular pipes of sheet iron, about $\frac{1}{16}$ inch thick, are used instead of wood; these are bolted together by means of two flanges at each end. It is highly important that these pipes should be very carefully fastened, either to the shaft-sides or to suspending ropes or chains, so that there is no possibility of their falling on the sinkers after being knocked by stones from the shots, etc.

Explosives.—For ordinary shales in the coal measures, gunpowder, either in grains or compressed, is found most effective in lifting the ground; but for hard stony ground or rock, dynamite, blasting gelatine, or other high explosive is found most effective. One of the advantages of dynamite is that, if the hole is bored as large as would be suitable for gunpowder, the charge can be concentrated at the bottom of the hole, and is then more effective for lifting the ground. Or, if a small hole is bored, it can be done more quickly than a large hole such as is required for gunpowder. A dynamite cartridge will go in a hole 1 inch in diameter. For gunpowder the hole should be 2½ or 3 inches in diameter, in order to take a sufficiency of powder.

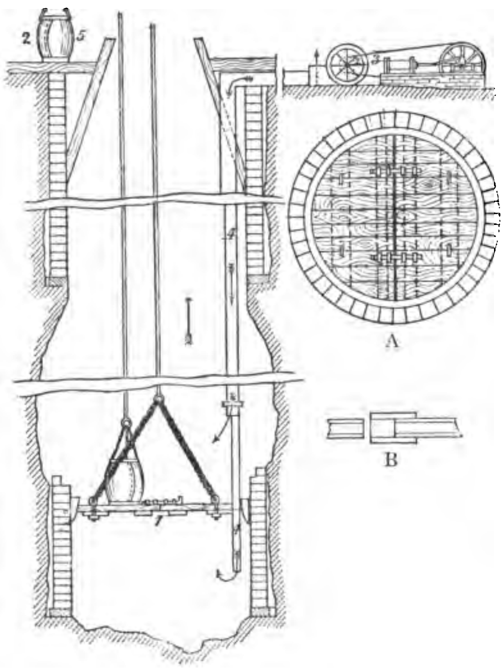


FIG. 171.—Ventilation and bricking of shaft. 1, bricking scaffold; 2, sinking bowks; 3, blowing-fan; 4, wooden air-pipes; 5, landing-stage; ←——, direction of air-current. A, plan of shaft, showing scaffold; B, section of joint of pipe.

Drilling by Steam-power.—In ordinary colliery sinkings the blast-holes are drilled by manual labour, because, though the holes could be more quickly drilled with a machine worked by compressed air, yet the saving in time in actual drilling would be lost in the time required to fix the machine, and the accidents connected with the use of machinery; but if the sinking is to go through a great thickness of hard rock, then steam-power is required.

It would be dangerous and inconvenient to take steam into the pit-bottom, and therefore the steam-engine is placed at the top, and used to compress air. This is conveyed to the pit-bottom in pipes from 2 inches to 4 inches in diameter, according to the number of machines to be worked. At the sinking of the Harris's Navigation Colliery, in South Wales, several descriptions of machine-drills were employed; the diamond drill was first tried, but subsequently it was superseded by the use of a percussive drill, similar to that shown in Fig. 172. This kind of drill is generally carried on a tripod stand. Details of a machine-drill are shown in Fig. 173. The machine consists of a cylinder 3 inches to 4 inches in diameter, with a piston having a stroke of say 4 inches; the slide-valve C reverses by the action of the piston D, so that the piston will move with great rapidity, say 500 double strokes a minute. The piston-rod projects through a stuffing-box, and at the end is a socket or drill-holder, S. The cylinder is fixed on an iron slide, up and down which it can be moved by turning a screw, R.

FIG. 172.—Rock-drill.

When proceeding to drill a hole, a short drill is fixed in the piston-rod, and light blows are struck upon the surface of the rock until a hole has been gently and slowly bored to the depth of an inch or two; this care in starting is necessary because the drill might slip on the bare surface of the rock; after once the hole is formed in the right place, full power is turned on. The drill turns itself between each blow (see Fig. 173). A fluted rod, F, projects through the cylinder cover into the cylinder, and fits into a fluted hole in the piston; the flutings have a spiral twist like those of a rifle. As the piston moves up and down, it tends to twist backwards and forwards to the fluted bar. This bar outside the cylinder has on it a ratchet-wheel, which permits it to turn in one direction only, so that the piston has to twist when making the back-stroke. In this way the piston is turned one tooth of the ratchet-wheel every back stroke, whilst the fluted rod turns every forward stroke. The turn of the piston, and consequently of the boring-bar, causes the tool to drill a round hole.

As the drill works, the miner turns the handle of the advancing screw, and so causes the cylinder to move down the slide, thus keeping the point of the drill up to its work. The slide being say

18 inches long, the hole can be drilled that depth. When this depth is reached, the screw is reversed, the drill drawn out of the hole, a longer drill is placed in the hole and cottered to the piston-rod. This second drill is rather narrower than the first drill, so

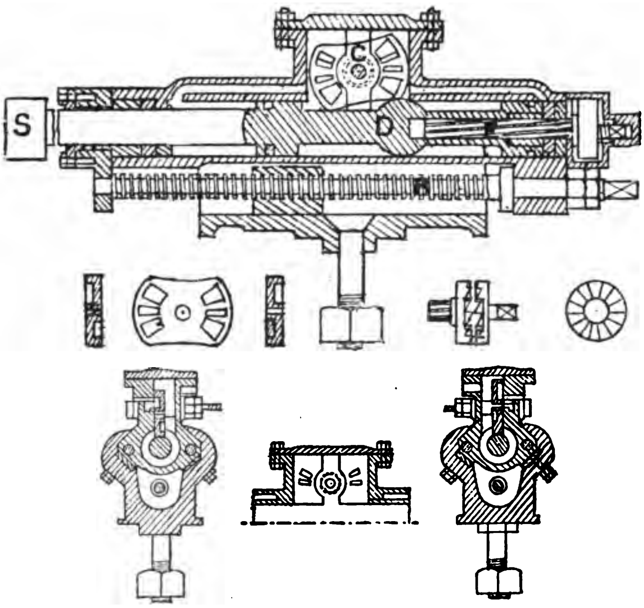


FIG. 173.—Rock-drill: details.

that it may not catch on the sides of the hole. When the entire length of this drill has been bored, if it is still required to go deeper, a third and fourth drill can be added. To clear the hole of dirt, a jet of water is directed into the hole, and the action of the drill mixes the dirt up with the water, some of which is always being jerked out of the hole. At Harris's Navigation they used Ingersoll's drills and Beaumont's drills. Ingersoll's drill cylinder had a diameter of 3½ inches, and a stroke of 4½ inches; three of these were worked at the same time. Ten holes were drilled, each 3 feet in depth, the sumping shots were put at an angle of 35° from the vertical, and were from 3 feet to 3 feet 6 inches deep, and 1½ inch in diameter at the top and 1¼ inch at the bottom. It took 5 minutes to fix the drilling-machine, and from 15 to 20 minutes to bore a hole 3 feet deep; in the same ground it took three men 2½ hours to bore one hole by hand. Two men attended to each machine-drill. By the use of machine-drills the sinking in the Pennant rock was made at the rate of 3½ yards a week, whilst by hand the progress was only 2½ yards a week; no time spent in walling included in either case. The diameter of the excavation was 20 feet. For every yard sunk at this pit in this rock, there were required 33⅓ lbs. of dynamite. The brickwork was 18 inches thick, and the diameter inside was 17 feet, and 1 yard high was built in 10 hours by six masons and four labourers.

As an example of the difficulty of sinking in hard rock, it may be mentioned that at this same sinking only 2 yards a week were sunk and walled when in hard rock, whilst 7 yards were sunk and walled when in ordinary shales. This sinking is one of the most interesting of which a record has been published. Details are given by Messrs. Forster Brown and Adams, in a paper read at the Institution of Civil Engineers, in 1881. The area of the royalty (that is, the mineral estate) to be worked by this colliery is 3500 acres, and the depth of the coal 760 yards. The two shafts were 60 yards apart, each 17 feet in diameter inside, and the total cost of the sinking and plant was £300,000. The time occupied in sinking was 6 years 3 months, divided as follows:—

	Per cent.
Proportion of time spent in actual sinking	50·5
" " " " walling	12·45
" " " stoppages (holidays, strikes, etc.)	18·3
" " " boring drill-holes	2·7
" " " making lodge-rooms (*i.e.* chambers in the shaft-side for pumps)	16·05
	100·00

In sinking shafts for metalliferous mines, machine-drills are generally employed, because of the hard nature of the rocks. As a general rule, machine-power drills should be employed wherever the rocks are very hard, and hand-drilling wherever they are soft, like common coal-measure shales and binds.

Speed of Sinking.—When there is a large quantity of water in a sinking pit, the speed of sinking is often very slow, owing to the time that has to be occupied in attending to the pumps, so that, even if they are entirely efficient, the sinking is much hindered. In some cases the influx of water is such that the sinkers are perpetually standing in it, often knee-deep, and it is obviously difficult to make rapid progress under such circumstances. Attempts are usually made, if the ground is at all solid, to tub or dam back the water. In the case of dry or nearly dry sinkings, the rate of progress depends chiefly on the organization and the capacity of the machinery. Thus in case there is only one winding-engine working, as is generally the case, with one rope, a good deal of time will be occupied in drawing the dirt after a blast. For instance, a shaft 15 feet in diameter has an area of 176 square feet; then, if 14 cubic feet weigh 1 ton, there are 12½ tons of material in every foot of depth, or 37½ tons per yard. If the hoppet is a large one, say 3 feet in diameter by 3 feet deep, it will hold about 15 cwt.; it will thus require 50 hoppets to take out 1 yard in depth of the shaft; and if it takes 4½ minutes to raise a tub—say, 1 minute winding up, 1 minute lowering down, 1¼ minute hooking on and steadying at pit-bottom, 1¼ minute unhooking and changing hoppets at the pit-top, total 4½ minutes—then it will take 225 minutes, or say 4 hours, to wind the material contained in 1 yard of strata, supposing that there is no hindrance of any kind; but hindrances must always be expected. If there are two winding-engines, then the work may be done in much less time. As much as 20 yards have been sunk (without walling) in a colliery shaft in one week; but 8 to 10 yards a week sinking and walling is good sinking in ordinary measures.

Bricks and Stones for Shaft-lining.—Where stone is plentiful, it is sometimes used for building the circular lining of the shaft. At collieries bricks are more common. The dimensions of an ordinary brick are 9 inches × 4½ inches × 3 inches; sometimes bricks of special shape are made, say 6 inches × 6 inches × 3 inches. Where the ordinary-sized brick is used, the wall is double, being 9 inches thick; about every fourth course are headers to bind the two rings of brick together. When rectangular bricks are laid in a circle, the internal corners touching, the external corners are some distance apart; to avoid this the bricks are often made in a mould curved to the circle of the shaft (see Fig. 174). In shafts of small size, 6 or 8 feet in diameter, this is an advantage; but in shafts over 10 feet in diameter it is not always easy for the masons to tell which is the shorter side of the brick, and he may lay it the wrong way, and for that reason rectangular bricks are generally used; the wider the brick, however, the greater the difference between the length of the front and back, if curved. Where 6-inch bricks are used, one ring would be used instead of two 4½-inch. When laying the bricks they must be sheltered from the water falling down the shaft, otherwise the mortar would be washed away. Hydraulic lime or quick-setting cement should be used in wet shafts.

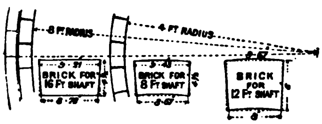


FIG. 174.—Circular bricks.

Permanent lining of shafts is sometimes made of wood and iron, as shown in Figs. 175 and 176. In this case the shaft is lined with wrought-iron rings about 8 inches deep, with an internal flange at the top and bottom; behind these are placed 2-inch planks; wedges are placed between the rings and the planks to fasten them. Iron cross-beams are fastened to the circular rings, by means of which the shaft is partitioned. The diameter of the shaft is about 15 feet 8 inches. When seen by the writer in 1883, the sinking was in progress, and it was intended to carry the shaft to a depth of 800 yards.

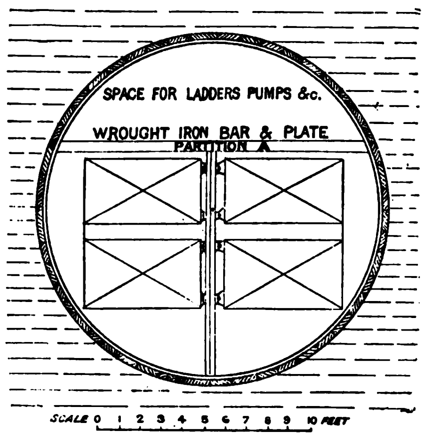


FIG. 175.—Plan of new shaft, Clausthal.

Tubbing and Coffering.—Reference has been made above to tubbing. This is the technical term for water-tight shaft-lining made fast to the strata at the bottom, so that the water cannot enter the pit. It may be made with brick, stone, concrete, wood, or iron; when made with brick or stone it is often called coffering. Cast-iron tubbing is the kind most frequently employed in England in colliery shafts. Fig. 177 shows a section of a hillside with a shaft sunk down to a seam of coal. About 40 yards below the surface is an offtake drift, or water-level, by which the ground is drained to that depth. The strata down to a depth of 50 yards consist of binds, shales, and fire-clay, through which water will not readily pass. Below this a bed of rock is sunk through; this rock, having many joints, and being also porous, is readily permeable by water, of which it contains a large supply, because it

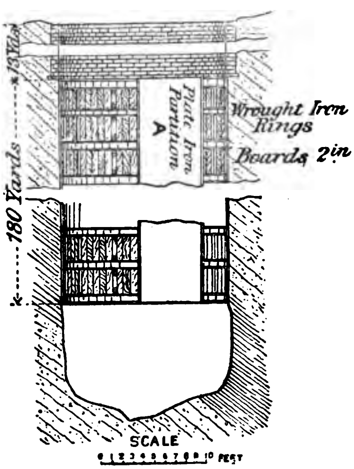


FIG. 176.—Section of new shaft, Clausthal.

crops out on the hillside, and the rainfall and water from brooks sink down into it, whence they pass through the rock into the shaft. The sinking was effected by means of pumps. Below this bed of rock were beds of shales impervious to water. About 5 yards below the bottom of the rock a wedging-curb was put in, and on this tubbing was built up to the offtake drift. The water was then excluded from the shaft up to and a little above the level where it was free to run off. Continuing the sinking, another bed of water-bearing rock was found. This rock cropped out on the hill-top, and yielded a copious supply of water. Before sinking through this rock, a wedging-curb was placed in the water-tight shale above it. After passing through the rock, which was about 15 yards thick, another wedging-curb was placed in the water-tight shales below. Tubbing was then built up from the lower curb to the upper one, and thus the water was excluded from the shaft. The sinking was continued for a further depth of 100 yards, where the coal to be worked was reached. The coal and the strata immediately above and below being dry, the works are also dry, and the expense of pumping is permanently saved by the tubbing in the shafts. In case, however, that instead of 100 yards of shales below the second rock and above the coal, there had only been 30 or 40 yards, it is probable that, when a considerable extent of coal had been got, the roof would have broken down and the water in the rock would come into the workings, causing great expense by its softening action upon the roof and floor and pumping required. Thus the expense of the lower length of tubbing would have been worse than useless, because it would be better to drain the rock by pumping at the shaft and have the workings dry.

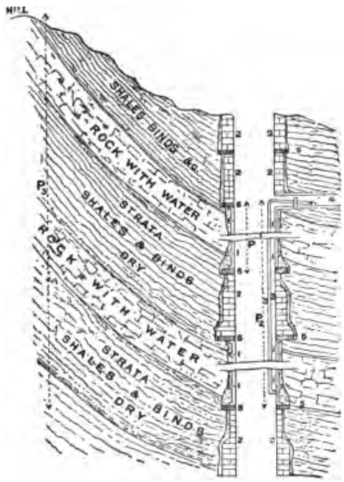


FIG. 177.—Section of shaft, showing brick-work and tubbing. 1, tubbing; 2, brick-work; 3, pipe to take pressure off lower tubbing; 4, offtake for water; 5, wedging-curbs; P, P_2, P_3, head of water exerting pressure against tubbing.

What thickness of water-tight strata is necessary can only be ascertained by experience in each particular district, because the results obtained vary greatly. In the South Wales Coal-field tubbing is rarely employed, whereas in North Wales and in

England it is very frequently employed. At a certain Yorkshire colliery, the coal, 4 feet thick, lay 60 yards below a heavily watered bed of rock, which was tubbed at the shafts. The coal was worked longwall, and there was plenty of dirt for making packs and stowing the goaf, and the water from the rock above did not come down in any large quantity. At another colliery the coal, about 5 feet thick, was about 60 yards below a water-bearing rock. The coal was worked pillar-and-stall; some packs were built, but not so many as in the first instance. Occasionally the water from the rock above, which was tubbed at the shafts, broke in, causing great expense. In some cases a thickness of 30 or 40 yards of shale are sufficient to keep the water from breaking through into the workings below, and in others it is probable that 100 yards would be insufficient. It is evident that the thickness required will depend not only on the nature of the shales, but also possibly upon the head or pressure of water in the rock, and upon the thickness of the seam of coal and the method of working. Thus if 2 feet of coal only were extracted, and the goaf tightly packed, there might be no fracture of the strata; whereas if 12 feet of coal were got, and no packs, there would be very serious fracture of the strata.

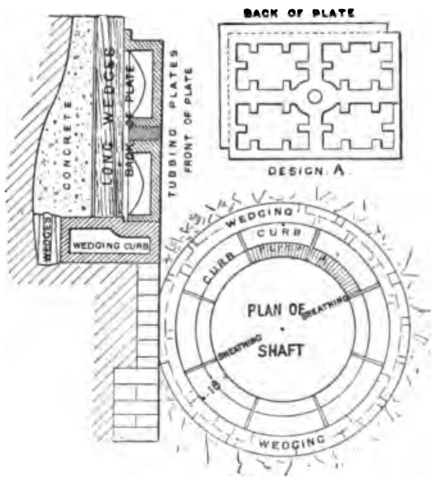


FIG. 178.—Cast-iron tubbing.

Cast-iron Tubbing.—Fig. 178 shows some tubbing in plan and section. A wedging-curb is a cast-iron ring varying from 9 inches to 3 feet in width, but seldom exceeding 2 feet, and commonly about 15 to 18 inches in width, and 7 inches in depth; it is hollow inside, the metal being about $1\frac{1}{4}$ inch thick. There is a rebate, or step, *a*, on the inner side $\frac{1}{8}$ inch wider than the width of the base of the tubbing-plate. If another length of tubbing has to be brought up to this curb from below, there is then also a rebate on the under side. The curb is cast in segments of convenient length, say 5 feet. Where the curb has to be placed the shaft is belled out, so as to leave a space 3 or 4 inches in width at least at the back of the curb. The wide excavation is carried sufficiently high to allow the free use of a hammer; the curb-bed is cut out with a pick axe, chisel, and wedge so as to avoid the shattering action of explosives. It is carefully levelled; a sump is left in the middle, in which the water-barrel may dip. The segments of the curb are placed in a circle; between each joint is placed a board of soft pine about $\frac{3}{8}$ or $\frac{1}{2}$ inch thick. In order that the curb may be placed round the centre, an iron pin is fixed in the middle of the shaft (in a beam fixed across the sump) by means of a centre-line and plummet. A rod called a trammel is fixed by a collar over this pin; the end of this rod is moved round to see that each segment of the curb is an equal distance from the centre; blocks of yellow pine are placed with the grain vertical all round the curb between it and the strata, and tightly hammered down. These blocks are then wedged. The wedges are made of pitch pine (see *a*, Fig. 179); they are 7 inches long, $2\frac{1}{2}$ inches wide, $\frac{1}{4}$ inch thick, and tapered to a knife-edge at the bottom. To make these wedges a resinous piece of pitch pine is selected, and cross-cut into blocks as thick as the length of the wedges; it is then split up, and the splinters carefully shaved and sharpened; these are then dried for some weeks or months, say in the boiler-room, and they become hard. These wedges are carefully driven in so as not to break them before they are driven up to the head. The wedging is done equally all round the shaft, the trammel being used to see that one segment is not driven nearer to the centre than the other; the blocks soon get too hard to admit wedges. Then a steel chisel (see *b*, Fig. 179) is driven into the blocks and extracted by the forked lever *c*; the wedge is immediately put in the hole thus made, and driven in. As the wood gets harder it is necessary to use a two-handed hammer, *d*, to drive in the chisel, and the man has to jump on the extracting lever to get the chisel out. At this stage the long wedges cannot be driven home, and shorter wedges, $4\frac{1}{2}$ inches in length, are then used. The opening chisel is put in by holding it with the left hand and striking it with the small hammer till it just sticks in the top of the wedging; it is then struck with a two-handed hammer; if, instead of going in, it springs out, the wedging is sufficiently hard.

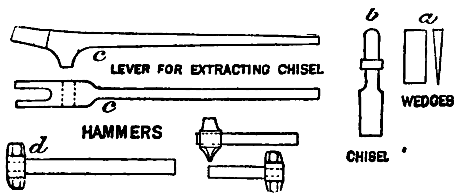


FIG. 179.—Tools used for tubbing.

It will take 72 hours, six men working at a time in 8-hour shifts, to wedge such a curb in a shaft 13 feet internal diameter. On the top of the curb the belled-out space is now filled up with brickwork set in cement or cement concrete, and grouted with cement all round, so that the wedging is securely covered up and kept down. The tubbing-plates, cast in segments of a convenient size (see Fig. 178), are now built up in a circle on the rebate of the curb; underneath each plate is a piece of board of yellow pine $\frac{3}{8}$ inch thick, with the grain pointing towards the centre of the shaft, and between each segment is a similar piece of board with the grain pointing towards the centre of the shaft. Long wedges are placed vertically behind the plates, the thick end of one wedge being put to the bottom, and then the other wedge driven down, liners being placed in the space if necessary; there will be a wedge at the centre of each plate and one at each joint. By driving in these large wedges the segments are squeezed tightly together. Some pitch pine wedges about 4½ inches long are now driven into the vertical joints to make them tight. The tendency of these wedges is to widen the circle, while the tendency of the long wedges at the back is to close the circle; if the circle is too much widened, additional wedges must be put behind. When the vertical joints are thus made tight, the holes behind the tubbing may be filled up. This filling is sometimes done with any harmless material at hand, such as small shale, fire-clay, etc.; sometimes cement concrete is put in; other engineers prefer to fill up with finely riddled garden soil. The next ring of tubbing is now placed on the top of the first, wooden sheathing being placed in all the joints as before, the vertical joints of the second ring being in the middle of the plates of the lower ring; long wedges are driven in behind, and short wedges in the vertical joints as before, and the operation repeated until the tubbing has been carried up to the water-level or to some wedging-curb. When the tubbing stops at an upper wedging-curb, unless the length has been carefully calculated, it may be necessary to have a length of tubbing-plates cast to make up the length. It is difficult to put wedges in behind the top ring of plates when it ends with a wedging-curb. The space behind this last ring may be filled up with cement concrete, so that the plates cannot be driven back by the wedging. After the length of tubbing has been all fixed, the horizontal joints may be wedged (it is evident that it would have been useless to wedge these previously, as the plates would have been merely lifted). It will probably be necessary to go over the vertical wedging again, and both horizontal and vertical joints will have to be wedged two or three times before the wedging is water-tight. When this is done, the hole in the centre of each plate may be plugged with a round block into which small wedges are driven. It is usual to fix a tap in one of the plates of the tubbing, and to carry a small pipe, say 2 inches in diameter, up to the water-level. The object of this pipe is to relieve the tubbing from any excess of pressure. Whether or not it has any such effect may be doubtful; it serves, however, the purpose of indicating the pressure of water at the back of the tubbing, and also may be convenient in case a supply of water is required.

Design and Strength of Tubbing.—Design B (Fig. 180) is a very good one. The dimensions are given on the figure, which shows a back view of the plate and the section. When the plate is fixed in the shaft the ribs or flanges are not visible, the front of the plate being quite smooth. The strength of the plate to resist bending pressure lies in the ribs. The front plate connecting the ribs prevents the water entering the shaft; it will be seen that in a plate 2 feet 6 inches high there are five horizontal ribs, and in a length of 5 feet 2 inches there are five vertical ribs. These ribs are stiffened by brackets. The lower flange or rib is 5 inches wide, the upper rib is $5\frac{1}{2}$ inches wide, and has a flange about $\frac{3}{8}$ inch thick standing up at the back about $\frac{3}{4}$ inch high; this flange is continued down one side of the plate, and serves as a guide against which the next plate can be placed; it also prevents the wedging from driving the wooden sheathing back. In the centre of the plate is a hole through which the water can pass until the joints have been securely wedged; it is also convenient as a pin-hole for a shackle when lowering the plate down the pit.

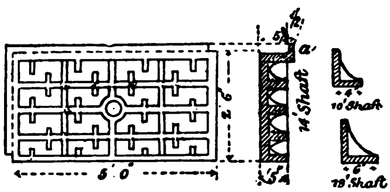


FIG. 180.—Tubbing-plate, design B, back elevation and vertical section.

Width of Ribs.—The following rules for the design are in accordance with actual practice: the ribs, or flanges, to be not less than 4 inches on the bed for a shaft up to 10 feet in diameter; $4\frac{1}{2}$ inches for 10 feet to 13 feet; 5 inches for 13 feet to 16 feet (see Fig. 180); $5\frac{1}{2}$ inches for 16 feet to 19 feet; 6 inches for 19 feet to 22 feet. These dimensions, 4, $4\frac{1}{2}$, 5, $5\frac{1}{2}$, and 6 inches, are values of a (see Fig. 180, and also p. 128).

Thickness of Plate and Ribs.—The following is a rule given by Greenwell for the thickness of the plate.[1] The ribs, or flanges, are all made the same thickness as the plate. Let D equal the diameter of the shaft in feet, and P equal the depth in feet below the water-level, x equals the thickness of the plate in feet;

[1] This rule is much quoted by writers; it is, however, obvious that to be useful it must be combined with other rules as to design and width of ribs, etc.

then $\frac{P \times D}{50,000} + 0{\cdot}03$ foot $= x$. If P = 300 and D = 14, then $\frac{300 \times 14}{50,000} + 0{\cdot}03 = 0{\cdot}114$, which is the thickness of the plate in feet. This multiplied by 12 = 1·368 inch, or say $1\frac{3}{8}$ inch. If the plate is made on design B, with the width of ribs above given, and with the thickness of the plate and flanges according to this rule, and properly put in, it will never burst. Another design, A (Fig. 178), is similar to B, except that there are only three ribs instead of five, and the plate is therefore a little lighter; for very shallow depths the thickness given by the rule is too light, as it might be accidentally broken. For these shallow depths, therefore, the plates may be made thicker, and design A employed; in this case the thickness of the plate should be 50 per cent. stronger than that given by Greenwell's rule. This rule is useful for rough-and-ready calculations, but it is evidently not very accurate, because the thickness of the plate varies directly as the depth, while the strength to resist bending given by increased thickness is more than in direct proportion to this thickness.

It is evident that tubbing made according to the above rule will not give the best possible disposition of the iron, in the case of deep shafts, where the tubbing is 200 yards and upwards in depth, and that the right way would be to increase the width of the flanges. Thus if a rib, or flange, is 5 inches wide (the thickness of plate is in all cases included in the width of the rib)—and that is a suitable width for a depth of 200 feet—then for a depth of 600 feet the width of the rib ought to be increased. If the ribs are regarded as girders, their strength will follow in proportion to the square of their width; thus a rib 8 inches wide will be four times as strong as a rib 4 inches wide; but if the ribs and plate are regarded merely as the stones of an arch, then their power of resistance will be in direct proportion to the weight of metal in the plate and *horizontal* ribs. It is best to take a middle course between these two points of view. If, therefore, the thickness of the plate and ribs is increased in proportion to the *square* root of the pressure, and the width of the flanges also in proportion to the *cube* root of the pressure, we shall get a plate more calculated to resist the variety of strains than if the thickness only is increased directly as the pressure. Thus a rule might be made as follows:—

x = thickness of plate in *inches* for a depth of 200 feet.
P = depth in feet.
D = diameter of shaft in feet.
y = thickness of plate for depths greater or less than 200 feet.
z = width of ribs for depths greater or less than 200 feet.

a = width of rib for 200 feet depth (see Fig. 180). (See also first rules for design, p. 126.)

$$x = \frac{P \times D}{4000} + \tfrac{1}{3} \text{ inch; or, since } P = 200, \text{ then } x = \frac{D}{20} + \tfrac{1}{3} \text{ inch}$$

$$y = x \times \sqrt{\frac{P}{200}} \qquad z = a \times \sqrt[3]{\frac{P}{200}}$$

The following are the width of ribs (including front plate, or web), and the thicknesses of plates and ribs by the above rules for a 14-foot shaft:

Depth.		Thickness		Width of ribs.
100 feet	...	0·72 inch	...	4·00 inches
200 "	...	1·03 "	...	5·00 "
300 "	...	1·26 "	...	5·75 "
600 "	...	1·78 "	...	7·20 "
900 "	...	2·18 "	...	8·30 "

This rule for design B is suitable for all depths and diameters, and gives a very strong plate without undue weight, because of the width of the ribs.

In calculating the pressure, it is necessary to consider the height of the source of the water (see Fig. 177). In this case the water, pressing against the lower length of tubbing, comes in a rock that crops out on the top of the hill, and the pressure against the tubbing will be that due to the depth from the top of the hill to the bottom of the tubbing.

Faced Tubbing.—Tubbing has been made with internal

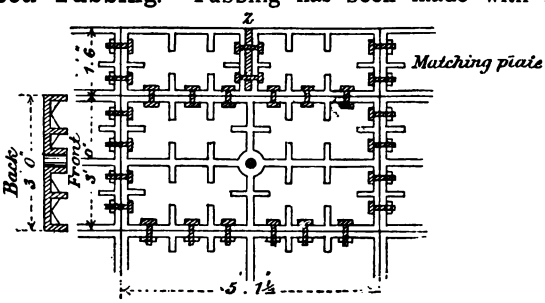


FIG. 181.—Tubbing-plate, inside flanges.

flanges (see Fig. 181), the vertical joints planed and the segments united with bolts, the entire ring of plates being then placed on a lathe and turned; in this way a whole length of tubbing may be accurately fitted. Bolts unite the horizontal joints. A layer of soft felt soaked in tar, ⅛ inch thick, is placed in each joint; when the bolts are tightened, this felt is squeezed, so that the plates seem to touch.

In putting in tubbing of this kind, wooden sheathing is placed on the horizontal joint on the rebate of the wedging-curb, and also on the top of the highest ring of tubbing, and these two are the only joints that require to be wedged. The plates, being all numbered and marked, can be put together in the pit as quickly as they can be lowered and with very little labour. In the top ring the last plate is made in two pieces (see Fig. 181), and a 2-inch strip of iron is inserted in the middle; the sides of this strip and of the adjoining edges of the plate being parallel, the strip can be pushed in when all the other plates are in position. Bolts pass through this strip and through each of the adjoining flanges, and so it is held securely. The advantage of this kind of tubbing is the rapidity with which it can be placed in position, and its strength, because of the support that each plate gives to the next one through the bolts, and there is no strain on the tubbing as the result of wedging, but simply that due to natural causes. It is probable, however, that this bolted tubbing will be more rigid than wedged tubbing, and therefore less likely to yield to any movement of the ground; on the other hand, if it does crack, the fragments will not fall out.[1]

Against the advantages of this species of tubbing must be put the extra cost of machining the joints.

In order to provide for pumping and other machinery, it is necessary in every long length of tubbing to put in wall-boxes (see elevation and section, Fig. 182) or rings with internal flanges; cross-beams can be rested in these wall-boxes. The depth should be about 2 feet, and the width of the wall-boxes inside the plate 10 inches. The vertical ribs are shown 1 foot apart, with intermediate brackets to prevent the flanges being broken by the pressure of the wedging. For very heavy machinery wider wall-boxes must be put in, resting on and securely fastened to the solid ground.

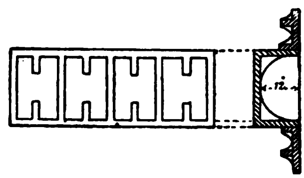


FIG. 182.—Wall-box, 5 feet × 2 feet, 12 inches deep.

Coffering.—Fig. 183 shows a cast-iron wedging-curb on which is built an internal brick wall 9 inches thick laid in cement, and an external brick wall $4\frac{1}{2}$ inches thick. Between these walls is poured a mixture of sand and cement, making a water-tight wall. A modification of this is to use stones instead of brick.

[1] The writer believes he was the first to introduce this kind of tubbing, at a colliery in North Wales, in the year 1873; it is still perfectly sound.

Another modification is (see Figs. 184 and 185[1]) to build a brick wall 14½ inches thick, of three rings each 4½ inches and two spaces each ½ inch, these spaces being grouted with cement, the middle 4½-inch ring having its courses half a brick higher than the external rings, so as to break the horizontal joint. Another method described by Mr. Embleton, in his paper to the Midland Institute, is shown in Fig. 186, which gives the dimensions. This stone tubbing consisted of ashlar, carefully dressed like the stones forming the arch of a bridge, laid in cement, forming a wall 9 inches thick. The stones of every fourth layer were perforated with one hole in the centre to let the water through; as the walling was built up these holes were filled with wooden plugs. The depth to the bottom of the tubbing was 44 yards 1 foot 3 inches, and the total cost amounted to £10 16*s*. per yard. The space at the back of the stone wall was filled with riddled soil. After the wall was completed, the shaft was quite dry, and a hole bored through one of the wooden plugs proved that the soil kept the back of the wall quite dry. A wedging-curb is always the foundation of this coffering.

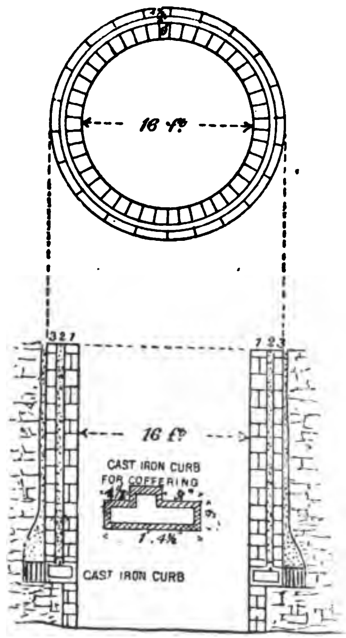


FIG. 183.—Brick-and-cement coffering. 1 brick 9 inches thick; 2, cement 3 inches thick; 3, brick 4½ inches thick.

Wooden Tubbing.—Wooden tubbing (see Fig. 187) is commonly used in France. It is made of wooden blocks sawn out of the best heart of oak. Each block is 3 feet long and 9 inches square; the joints are cut in radial lines, so that, when placed in the shape of a dodecahedron, the blocks fit together and cannot be pressed inwards. Each block is carefully planed, so that the joints are quite close. The curb is wedged in a different manner to English wedging-curbs. It is made of wooden blocks like the rest of the tubbing; the space between the block and the ground, being say 6 inches or more, is partly occupied by

[1] "Coffering of Shafts," by W. N. Griffiths, N.E Institute Mining Engineers.

a piece of wood, say 2 inches thick; between it and the ground the space is filled in with compressed moss; between the second piece of wood and the curb, wedges are placed and driven in,

Fig. 184.

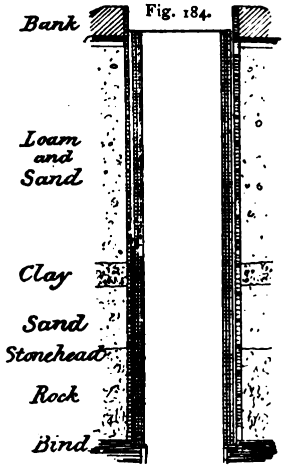
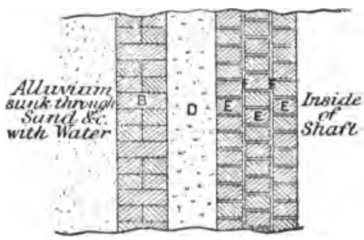
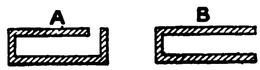


Fig. 185

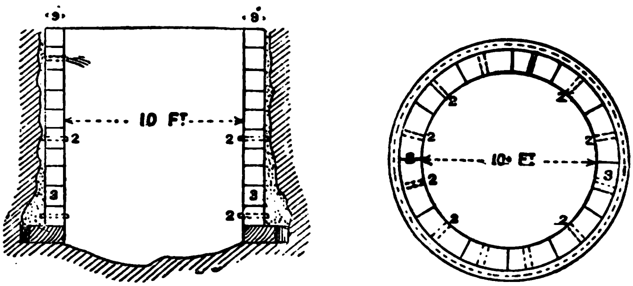




Sections of cast-iron wedging-curbs.

FIG. 184.—Brick-and-cement coffering. FIG. 185.—Section of coffering. B, back casing of 9-inch dry brickwork put in while sinking; D, puddled clay; E, three rings of brickwork in hydraulic mortar; F, two rings of hydraulic mortar grouted in.

which compress the moss; against the ground two or more curbs are wedged, and the space at the back of the blocks is filled up. When all the blocks have been placed in position up to the top, they finish underneath a brick or stone wall going up to the surface.



Elevation of stone walling. Plan of stone walling.

FIG. 186.—Stone coffering. 1, concrete backing; 2, holes to relieve water-pressure; 3, stone walling.

On the top of the last ring of blocks is placed a series of short vertical screws working in nuts like a screw-jack. The screws all round the shaft are tightened simultaneously, so that all the tubbing is tightly pressed down. The space above them is then filled up with blocks carefully cut to fit, the screws being withdrawn in turn. The tubbing is now caulked, like the caulking of a wooden ship: the joints are opened with a chisel to a depth of 1½ inch, and into this hemp steeped in tar is forced with a caulking-tool. The late Sir W. W. Smythe described an instance of wooden tubbing put in at a depth of 524 feet. The French engineers prefer wooden tubbing, but the late Professor Callon, of Paris, said that in very deep mines iron was to be preferred, owing to the difficulty in getting timber of sufficient strength. It is probable that, if there is any movement of the strata, wooden tubbing would be more easily repaired than cast-iron tubbing.

Quicksand.—The ordinary operations of sinking are often

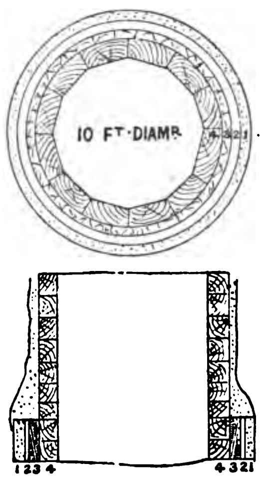


FIG. 187.—Wooden tubbing.

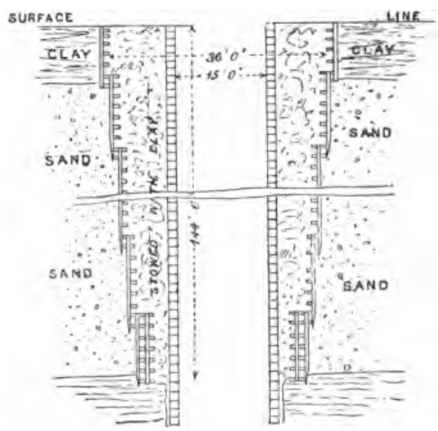


FIG. 188.—Sinking through quicksand by piling.

impeded by quicksand, which runs into the shaft like water, so that it is impossible to make it any deeper. This difficulty is sometimes overcome by a system of piling (see Fig. 188). In this case a wooden ring or curb of oak about 9 inches wide, 6 inches thick, and say 30 feet in diameter, is laid on the ground. Outside this and round it, piles, which consist of deal planks sharpened at one end, about 9 inches wide and 3 inches thick, are driven vertically down as deep as possible without breaking the piles. The ground inside the piles is now excavated, and an additional curb is placed inside, say 2 or 3 feet below the first. As the excavation proceeds, other curbs are placed inside the piles to prevent them being squeezed in. When the excavation has proceeded to a depth of say 3 feet less than that of the piles, another curb is laid inside the last, of less diameter, just leaving space for a row of piles to be driven down between the two curbs; another row of piles is now driven down all round, and the excavation proceeded with, internal curbs being placed in these as in the length above. The diameter of the shaft is rapidly contracted, as each set of piles reduces the diameter about 2 feet. This method of piling will not answer if the sand is very quick.

In order to keep the shaft the same width, the piling system has been modified in some places, as shown in Fig. 189. In this case the piles are driven at an angle of about 40° from the vertical on each side of a square frame. The shaft is then deepened, another square frame put in 2 or 3 feet below the first, and another set of slanting piles driven in, and so on till the solid ground is reached. This system also is inapplicable where

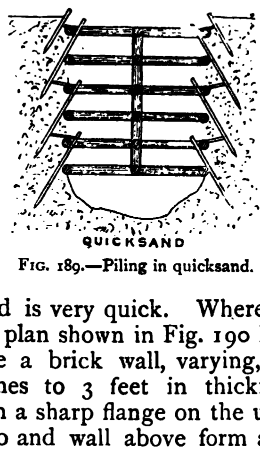


FIG. 189.—Piling in quicksand.

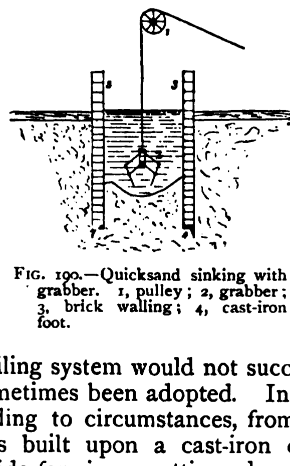


FIG. 190.—Quicksand sinking with grabber. 1, pulley; 2, grabber; 3, brick walling; 4, cast-iron foot.

sand is very quick. Where the piling system would not succeed, the plan shown in Fig. 190 has sometimes been adopted. In this case a brick wall, varying, according to circumstances, from 18 inches to 3 feet in thickness, is built upon a cast-iron curb with a sharp flange on the under side forming a cutting edge; the curb and wall above form a circular shaft; the wall is built up above the surface of the ground. The interior is then excavated, and the weight of the wall forces the cutting edge into the ground. The internal excavation is carried on as deep as is practicable, the weight of the wall causing the cutting edge to be several feet deeper than the excavation. As it sinks, the length of wall is increased by building on the top. The descent of this circular brick wall is sometimes guided by wooden guides on the surface, and sometimes steadied by ropes and screws attached to it. If the cutting edge is a sufficient distance below the excavation, the sand may not run in so quickly as to stop the sinking

But in order that men may work on the excavation, it is necessary to pump out the water. This pumping causes the sand to flow rapidly, and may make it impossible to continue the work.

In order that the work may be done without pumping the

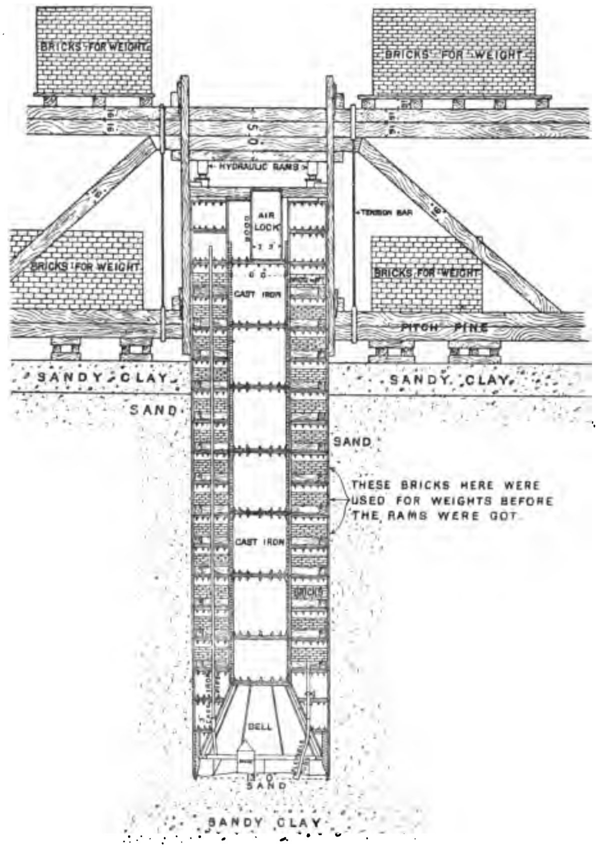


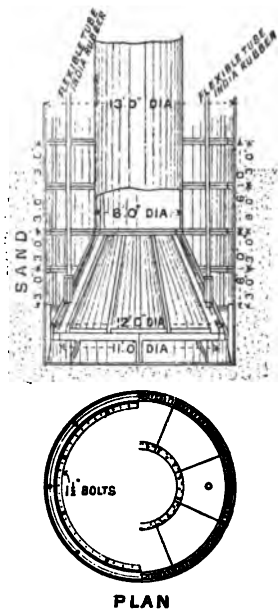
FIG. 191.—Pneumatic sinking, showing section of shaft during application of hydraulic ram.

water, it is necessary to send divers down who can work under water. With the shaft full of water the sand does not run.

In sinking through sand, this method presents no great difficulty if the depth does not exceed 80 feet. But if boulders are met with, and the depth exceeds 80 feet, considerable difficulty is sometimes experienced.

Fig. 190 shows a grabber, or mechanical excavator, employed to excavate the sand whilst the shaft is full of water. This method has been attempted, but with what success the writer does not know.

Another plan is the pneumatic system, which was carried out by the writer at the Bettisfield Colliery, in North Wales, of which the late Mr. J. T. Woodhouse was the consulting engineer.



PLAN

FIG. 192.—Bell air-cylinder and tubbing.

Fig. 191 shows a section of a shaft made of cast-iron plates bolted together by means of internal flanges, the joints wedged to make them water-tight. At the bottom of this shaft, which was 13 feet in diameter, was a cast-iron cutting edge, shown also in Fig. 192. On to a flange or shelf of this cutting edge was built a bell, reducing the diameter of the shaft to 6 feet; upon these were built cast-iron tubes 6 feet in diameter. At the top of this internal tube was the air-lock, as shown in Figs. 191 and 193. This air-lock had two trap-doors—one opening downwards communicating with the atmosphere, and the other opening into the 6-foot tube.

Compressed air was blown in through the tap A (Fig. 193) into the 6-foot tube. The artificial pressure thus produced forced the water in the shaft out again, through the sand, or by the 3-inch draining-tubes. The water being out, a man could enter the air-lock by the trap-door on the top. It was then closed. By opening a tap, he admitted air from the 6-foot tube into the lock. When the pressure in the lock was raised to that in the tube, he could then open the side door and leave the lock, when he would be lowered down to the bottom of the shaft by the windlass shown in Fig. 194. Materials could be sent in, and sand and other excavated stuff be sent out, through the air-lock without reducing the pressure in the bell. Whilst the pressure of the air forced the water out, it also tended to lift the bell. In order, therefore, that this might descend, it was necessary to let the air-pressure out, when the weight of the cast iron would cause the whole cylinder to descend. In order to increase this weight, bricks were placed in cross-bars between the 6-foot cylinder and the 13-foot cylinder, as shown in Fig. 191.

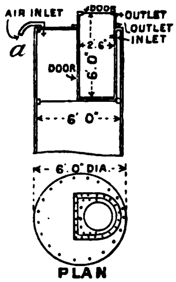


FIG. 193.—Air-lock.

After lowering the cylinder say 2 feet, the air-pressure would be blown in again, forcing the water out through one of the pipes provided, as shown in Figs. 191 and 192, and the work of excavation resumed. In passing through some clay and gravel, the cylinder stuck, and six hydraulic rams, capable each of applying a pressure of 60 tons, were applied to force the cylinder down, as shown in Fig. 191. The rams pressing upwards against the large frame, it was kept down partly by the weight of bricks placed upon it, and partly by attachment to piles, as shown in Fig. 194. Upon reaching solid ground, a wedging-curb was placed, and plates were carried up from the wedging-curb to the under side of the shelf-plate, to which the bell was attached. When the water-tight joint was thus completed, the 6-foot cylinders and the bell were removed, and a sound and dry shaft was completed.

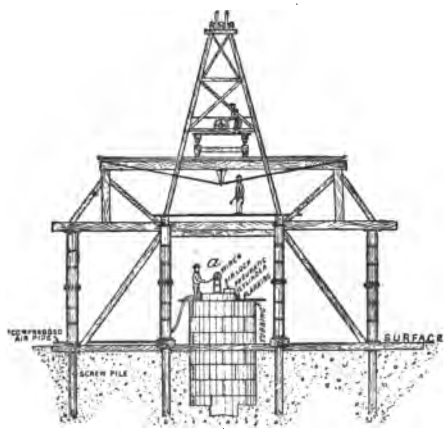


FIG. 194.—Surface arrangement.

The sinking of cylinders by the pneumatic process has been frequently adopted for bridge foundations and other purposes. In such cases, however, the cylinder is simply taken down a sufficient depth, and then filled with concrete. In this case a water-tight joint had to be made with the strata at a depth of 102 feet, involving a pressure of four atmospheres, or 45 lbs. above the ordinary pressure.

This operation is probably the only one of the kind yet performed in the mines of this country.

In working under pressure, great care has to be taken to avoid ill effects to the workmen. They must be physically sound, and well-fed. At a pressure of say 15 lbs. they may work four-hour shifts; at a pressure not exceeding 30 lbs., three-hour shifts; and, exceeding that pressure, two-hour shifts. Great care must be taken to avoid catching cold, especially in coming out of the air-lock; they must avoid the use of intoxicants. This is probably the cheapest way of sinking through quicksand when it is met with near the surface.

Sometimes a bed of quicksand is met with unexpectedly at

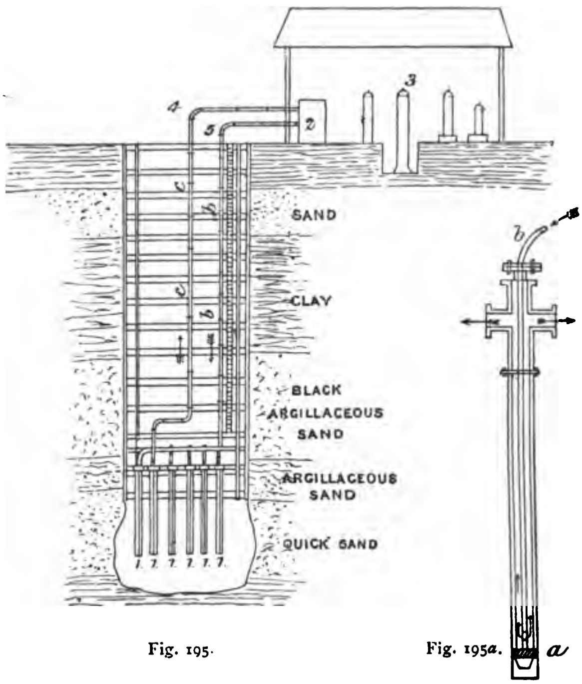


Fig. 195. Fig. 195*a*.

FIG. 195.—Poetsch's system of sinking. 1, freezing-tubes; 2, 3, ammonia process; 4, return-pipe; 5, feed-pipe. FIG. 195*a*.—Enlarged view of freezing-tube.

a considerable depth below the surface, where it is exceedingly difficult to use any of the systems above described. A very ingenious method has been devised and put in practice by M. Poetsch. His method consists in freezing the water and quicksand into a solid rock. Figs. 195, 195*a*, and 196 show the mode of carrying out this system. This process has been applied at a number of pits in Germany. The following table, No. VII., is copied from a paper by M. R. de Soldenhoff, printed in the "Proceedings of the South Wales Institute of Engineers," vol. xv. pt. ii., from

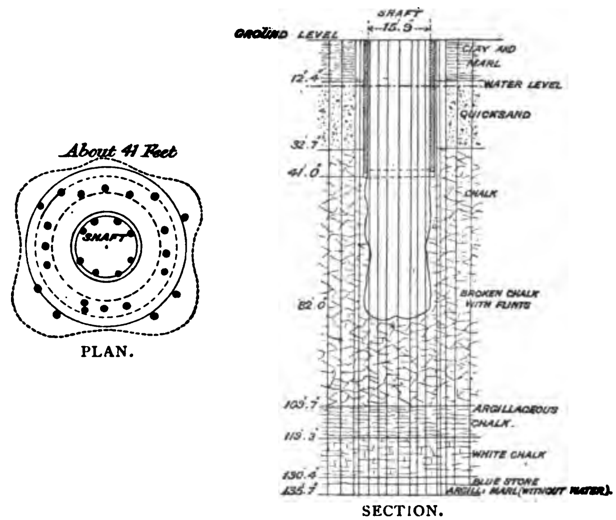


Fig. 196.—Poetsch's system of freezing bore-holes. Plan: black dots show bore-holes; volume comprised within the outer curved line to a depth of 135 feet 7 inches is about 149,030 cubic feet. Section: the black lines represent bore-holes.

which it will be seen that the process has been applied to beds of quicksand 100 feet in thickness :—

TABLE VII.

Names of pits where the process has been applied.	Thickness of strata above the quicksand.	Thickness of quicksand.	Number of days occupied in freezing the quicksand.	Form of the shaft	Size of shaft.	Total cost of freezing.	Cost per yard.
	ft. in.	ft. in.			ft. in. ft. in.	£	£ s. d.
Archibald ...	106 7	18 0	20	rectangular	11 4 × 15 6	1056	176 2 8
Centrum ...	14 10	100 0	57	rectangular	6 6 × 13 0	2000	60 0 0
Emilie I. ...	29 7	95 4	20	circular	7 8 diameter	2000	62 19 3
Emilie II. ...	29 7	95 4	30	elliptical	7 9 × 13 2	2000	62 19 3
Houssa ...	196 4	39 4	unknown yet	circular	16 6 diameter	1680	135 2 6[1]

[1] Practically, the thickness of the solidified body was 69 feet instead of 39 feet, and in that case the cost per yard would be reduced from £135 2s. 6d. to £73 1s.

The Archibald Pit was a rectangular shaft 19 feet 6 inches × 13 feet, the ordinary strata being 106 feet deep, below which was 18 feet of quicksand. Twenty-three freezing-tubes were sunk into the quicksand; each tube was of wrought iron, 8 inches in diameter, and when sunk to the required depth was stopped up at the bottom with a plug, as shown at *a*, Fig. 195*a*. This plug was composed of several layers of lead, cement, and pitch, so as to make the bottom quite water-tight. Inside each of these tubes was an internal tube (*b*, Fig. 195*a*), connected with the main cold-water tube (*b*, Fig. 195). The external tube was connected with the main return tube (*c*, Fig. 195). When all these twenty-three tubes were fixed and connected, a freezing solution was forced down the tube *b*, and so into all the internal tubes. This, escaping at the bottom, returned up the larger tubes and back up the main return tube *c* to the freezing apparatus. In this way the temperature of the tubes was reduced to about 25° Centigrade below freezing, the temperature of the returning fluid was 19° Centigrade below freezing, and all the water near was gradually frozen, so that in about thirty days the entire mass was frozen. The sinking was then carried on as if in rock. The solution that is used is water in which chloride of calcium has been dissolved, making a strength of 40° Baumé. This solution freezes at a temperature of 40° Centigrade below freezing, or about 40° below zero on the Fahrenheit scale. The freezing is produced by means of the Carré, or ammonia process. It is evident that, whilst this system has been successfully employed in several places, there may be difficulty with it in others, for the following reasons: If the water is pumped out of the shaft, then the quicksand will rise up, if it is really quicksand, and fill the shaft to a considerable depth; and if the pumping of the water continues, and there is a stream of water, it may take a long time to freeze the bottom of the pit. If, on the other hand, the water is not pumped, and the shaft is allowed to get full of water which is stationary, the water in the shaft will impart heat to the cold pipe, and so prevent the freezing, unless this cold pipe has been previously covered with a non-conducting material to keep the solution cold.

In 1892 a sinking by this process was made at the Lens Collieries, in the north of France, of which a description was given in the Federated Institute, by Mr. N. R. Griffith. At this place (see Fig. 196) the strata was soft and water-bearing, not quicksand, and it was desired to freeze it to a depth of 137 feet below the surface. The shaft was sunk in the ordinary way to a depth of 82 feet, when the water overpowered the pumps. Freezing-tubes were then put down in bore-holes, in the shaft-bottom, to the desired depth; freezing-tubes were also placed in bore-holes round the shaft. These holes were lined with sheet-iron casing $\frac{1}{8}$ inch thick. At the top they were 14 inches in diameter, decreasing to $10\frac{1}{4}$ inches in diameter, and finally to 8 inches in diameter at the bottom. The freezing-tubes placed in the bore-holes were flush-jointed, wrought-iron tubes, $4\frac{1}{2}$ inches in diameter, closed at the bottom; and inside each of these tubes was another wrought-iron tube $1\frac{1}{4}$ inch in diameter, with two slotted openings opposite to each other, at the bottom. The number of bore-holes in the shaft was eight, and those outside twenty in number, all the bore-holes being within a circle of about 35 feet diameter; but the freezing effect was supposed to extend 3 feet further on each side, making a total diameter of about 41 feet. In this case the water in the shaft stood level full during the operation of freezing. The temperature of the freezing fluid, which was a 20 per cent. solution of chloride of calcium, was 12° Centigrade below freezing as it left the cooling cistern, and 9° on its return, so that the temperature at the bottom of the bore-holes would probably be about 11° Centigrade below freezing. The freezing operation at this place was calculated to take 120 days.

One of the difficulties in the use of this system is the maintenance of perfectly water-tight tubes and joints, because an escape of the freezing solution permeates the ground with an uncongealable solution. To get over this difficulty, M. Gobert has designed a modification of the process, which consists in sending the liquid ammonia itself down the internal tube, which would evaporize whilst in the tubes and produce the required cold, say 30° Centigrade below freezing. The greater cold produced by this process would lead to a saving of time. The ammonia after leaving the tubes is compressed and cooled in the ordinary manner.

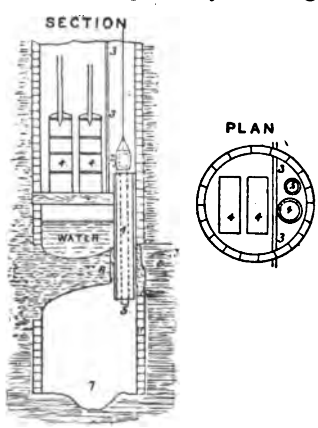


FIG. 197.—Lisbet's system of deepening a working shaft. 1, tube through which bowk can pass; 2, sinking-bowk; 3, brattice dividing shaft; 4, cages; 5, ventilating tube, *vide* plan; 6, concrete round both tubes; 7, sinking operations.

Deepening of Shafts.—It frequently happens that it is desirable to deepen the shaft whilst it is being worked in an upper seam. One method of doing this was adopted at the Lieven Colliery, in the north of France. In this case two holes were bored through the shaft-bottom, and a tube fitted into each, as shown in Fig. 197. The larger of these holes has a sufficient diameter for the sinking hoppet, the smaller for ventilation. Workmen sent down the larger tube enlarge the hole to the dimensions of the shaft exactly under the shaft above. The sinking then proceeds without interruption to the working shaft, above which it is bratticed so as to separate the sinking hoppet from the cages; at the same time, the water from the shaft above cannot get into the sinking below.

Another method is shown in Fig. 198. In this case an inclined road is driven from the workings to a chamber made under the sump of the working shaft. In this chamber is erected an engine which may be conveniently driven by compressed air, though steam, electricity, or water might be used. By very careful surveying, or perhaps by bore-holes, the extension of the shaft is set out exactly in the right place, and the sinking then proceeds as if from the surface.

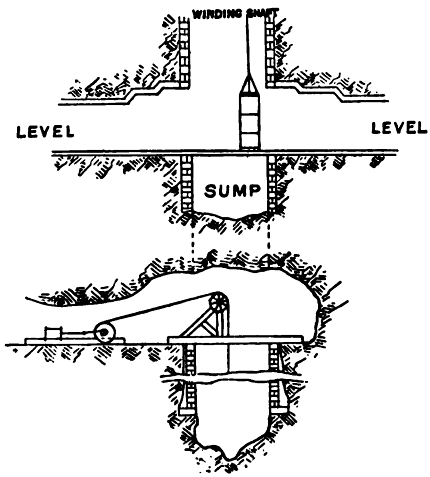


Fig. 198.—Method of deepening working shaft. Top levels communicate with lower workings by means of incline.

Kind-Chaudron.—It frequently happens in coal-mines that the upper strata are heavily charged with water, whilst the lower strata, in which the coal exists, are entirely protected from water by impermeable beds of shale. To take an extreme instance, at the Marsden Colliery, near Monkwearmouth, the pits were sunk through the magnesian limestone of the Permian formation (see Fig. 199). The shafts were not far from the sea and the limestone extended under the sea, and, being full of joints,

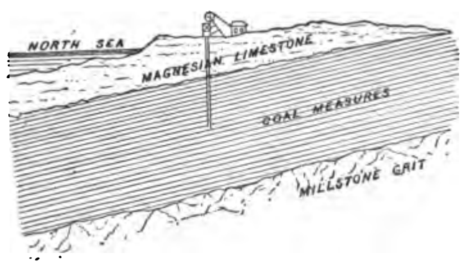


Fig. 199.—Limestone over coal measures (Marsden).

permitted the free access of sea-water. It was impossible to sink the shaft, though it was nearly filled with large pumps. In some mining districts, as, for instance, in the north of France and Westphalia, the coal-fields are overlaid with chalk (see Fig. 200), containing a very large supply of water, the coal measures below being perfectly dry. Therefore, whilst on the one hand enormous pumping machinery would be required to sink through the chalk in the ordinary way, when once the shafts were finished and properly tubbed no pumping at all would be required. To meet this contingency, Herr Kind, a well-known German engineer, and M. Chaudron, a well-known Belgian engineer, succeeded, in the year 1854, in devising a successful mode of sinking shafts through water-bearing strata, and of putting in water-tight tubbing down to the dry ground, without doing any pumping at all. The manner of doing this is as follows: The shaft is bored, as if it

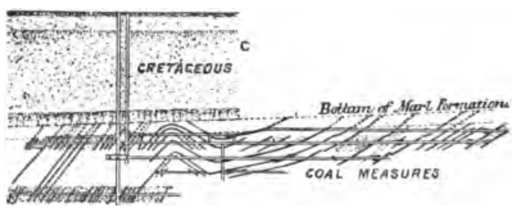


FIG. 200.—Chalk over coal measure (Westphalia).

were a bore-hole, by chisels fastened to boring-rods driven by a steam-engine in the manner already described for smaller borings. The arrangement of machinery on the top, though considerably modified, is somewhat similar to that shown in Fig. 77—one steam-engine working the boring-beam up and down, and another steam-engine winding the sludge-tank, or cleanser. It has been the usual plan to make the shaft by first boring a hole of small diameter, say 5 feet, and then enlarging it to the full diameter of say 16 feet. The boring-tool is called a trepan; that for the 5-foot hole is shown in Fig. 201, and weighs about 8 tons. It is made of forged iron,[1] and steel teeth are fitted in. The engine working the boring-beam has a cylinder about 40 inches in diameter, and 40 inches stroke, and makes say from eight to twenty strokes a minute. After working some time, it is necessary to send down a sludger, the same as in boring small holes. This is shown in Fig. 202. It is lowered down by a rope instead of rods. To enlarge the hole, a bigger trepan is used (see Fig. 203), weighing 16 tons. The *débris* from this large trepan falls into the

[1] Messrs. Krupp now make the trepan of cast steel, which they say is much stronger, thus permitting a greater drop, and faster work.

centre hole bored in advance, from which it is fetched up by the sludger. There is, however, a tendency for the dirt to consolidate. To avoid this difficulty, the hole is sometimes bored in three sizes (see Fig. 204)—the first hole say 4 feet in diameter, the second hole 5 feet in diameter, and the third the full size of the shaft. At the ledge formed at the bottom of the 5-foot bore-hole, a large iron tank is suspended into which the *débris* from the larger trepan falls. By sending down a hook the iron tank can be lifted and the *débris* drawn up. M. Lippmann has preferred to bore the hole from the beginning with one large trepan of forged iron with steel teeth (see Fig. 205), weighing 22 tons for an excavation 14 feet 2 inches in diameter. This trepan has one

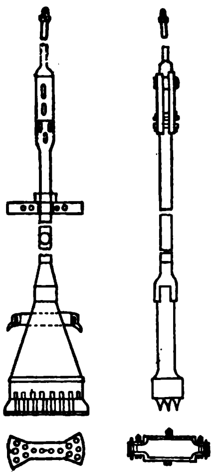


FIG. 201.—Small drill or cutting-tool.

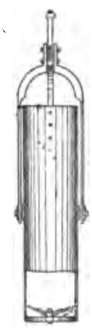


FIG. 202.—Sludger: Kind - Chaudron process.

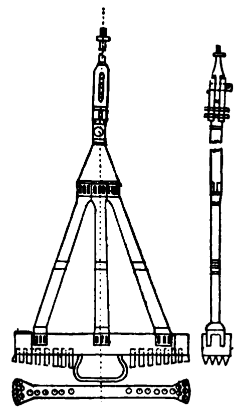


FIG. 203.—Large drill or cutting-tool.

row of chisels on each side of the centre near the middle of the pit, and two rows on each side near the circumference, which cut the ground into angular fragments. The boring is continued until the sound and dry strata are met with. The shaft being full of water, the sides stand firm without lining. In some cases a depth of over 200 yards has been reached by this process. The tubbing of the shaft is effected by means of iron rings or tubes cast in one piece about 5 feet in depth; the diameter is that of the shaft, with internal flanges. As these rings are too large for railway transport, a foundry has to be erected at the colliery. The joints are turned on a lathe. A thin strip of sheet lead is afterwards placed between the flanges, which are tightly bolted together, and the joints made water-tight by a caulking-chisel. As the tubbing is lowered down the shaft, fresh rings are bolted on at the top, until the whole length of say 200 yards is lowered to the bottom. As this length of tubbing, the lower rings of which are very thick, has an enormous weight, say 800 tons or more, no ordinary tackle would suffice to sustain the load. The weight has, therefore, to be borne by the water in the shaft, and a concave iron bottom is put near the bottom of the tubbing, as shown at *a* in Fig. 206. A tube rises up in the centre of the tubbing, through which rods can be passed. The interior being free from water, the tubbing floats, and the descent can be regulated to some extent by screws and chains. Before the tubbing is lowered into the shaft, a bed has been carefully levelled as shown at *c*, on

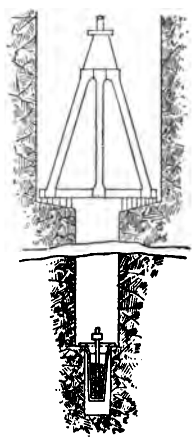


FIG. 204.—Three sizes of boring-hole.

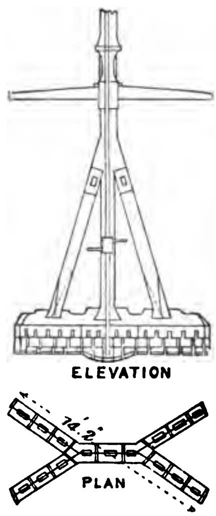


FIG. 205.—Lippmann's boring-tool.

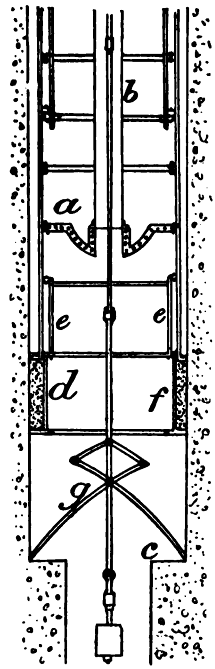


FIG. 206.—Mode of tubbing the Dahlbusch pit, Westphalia.

to which the tubbing is to be lowered. At the bottom of the tubbing is the most ingenious invention of M. Chaudron, called a moss-box, by which a water-tight joint is made with the strata at the bed *c*. The moss-box is made by forming the bottom ring of tubbing (*d*, Fig. 206) of smaller diameter, so as to slide inside the second ring; this second ring has the bottom flange external and not internal. The bottom sliding ring has its top flange internal, so that it may be suspended by bolts *e e*, and its bottom flange external. Between this flange and the bottom flange of the second ring is packed some dried moss, as shown at *f*, which is kept in its place whilst being lowered down the shaft by a covering of network. On arriving at the bottom, the whole weight of the tubbing above the moss-box has to be supported either by the water under the plate *a*, or by the moss *f*. If the shaft above and inside the tubbing is now filled with water, the whole weight of the tubbing comes upon the moss, compressing it and forcing it against the sides of the strata with such effect that a water-tight joint is made. In order to make the shaft still more secure against water, the space between the tubbing and the shaft-side above the moss-box is now filled in with mortar made of sand and cement, which is lowered to the bottom either in narrow boxes or in pipes; the whole of this space is thus filled up to the top.

In Fig. 206 is shown a forked tool, *g*, which has been sent down to scrape clean the bed *c*, before lowering the moss-box on to it. Having thus put in all the tubbing, the water can be drawn out of the shaft in barrels, and the internal tube *b* and the false bottom *a* removed; workmen can now go down and sink the shaft deeper. The use of explosives is now avoided, and the sinking is conducted with the utmost circumspection to avoid inrushes of water. After sinking say 6 to 10 feet further, an ordinary wedging-curb is put in, and the tubbing carried up to the moss-box; a little lower down another wedging-curb may be placed, and the tubbing brought up to the curb above.

CHAPTER VI.

"OPENING OUT;" OR, FIRST OPERATIONS IN THE MINERAL.

THE shafts must (by law) have not less than 15 yards of natural strata between them (this is sometimes increased to as much as 70 yards), and may be arranged as shown in Fig. 207.

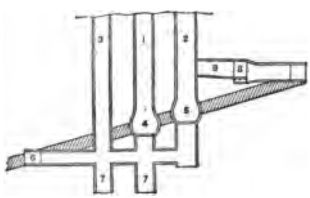


FIG. 207.—Section showing bottom of shafts. 1, downcast; 2, upcast; 3, pumping-shaft; 4, horse-level; 5, air-level; 6, water-level; 7, sumps; 8, furnace; 9, furnace-drifts.

The levels are driven either in the stone or in the coal, as the case may be.

In the case of a large colliery, the roadways immediately adjoining the winding-shaft will be very busy places, and it is necessary that they should be large and well protected. In many cases the levels adjoining the winding-shaft are arched like railway tunnels, the width varying from 15 to 25 feet, according to circumstances. The place adjoining the shaft is called the "porch." When the porch is arched, it is designed in one of the three ways shown in Fig. 208.

In Fig. 208 *a*, *a*, *a*, *a*, the arch is carried all round; the bottom part is called the "invert," the parts next above are called "side walls," and the top is the "arch." If the floor is sandstone, or any kind of stratum not easily softened by water, it is not necessary to have an invert. The side walls are curved, the better to resist the side pressure of the ground. The radius of the curve of the side walls is larger than the radius of the arch. The junction of the two curves must be imperceptible. To achieve this, the centre from which the curve of the side walls is struck must be on a straight line drawn from the bottom of the "arch" through the centre from which the curve of the "arch" has been struck, as shown in the figure. Where there is no invert, the side walls are generally built perpendicularly to a height of about 6 feet, from which the arch springs (see Fig. 208, *b*). The length of the porch may vary from 2 or 3 yards to 100 yards; it is generally higher adjoining the shaft (see Fig. 208, *c*). This extra height is convenient in the case of lowering long bars of timber or iron. It also facilitates the ventilation, for the two following reasons: Immediately over the workman called "hanger-on" at the pit-bottom is a wooden or iron roof, which serves as a shield to protect him from falling stone or coal, etc., and the air-passage should be above this shield; the hanger-on will thus be to some extent protected from the strength of the air-current and the dust carried with it. The air-current is less impeded by a sudden change of direction than would be the case with a low porch. There are also other reasons why a high porch is convenient; for instance, it gives space for pulley-wheels, steam-engines, and other machinery. In many cases the porch is not arched, but the roof is supported by wooden bars; and, in some

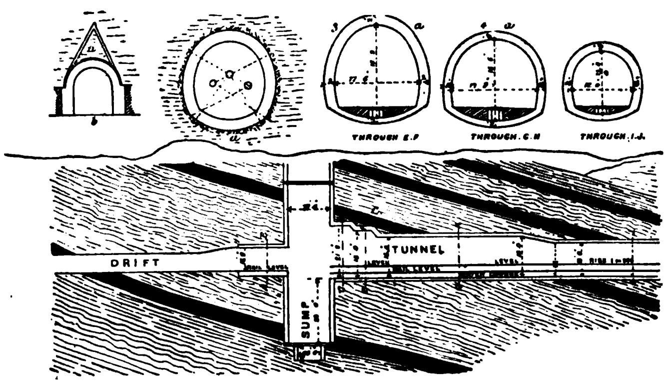


FIG. 208.—Cross-sections of arching.

cases, bars of iron, and more recently bars of steel, have been used. Occasionally it happens that no artificial support is required, a seam of coal, shale, or rock being sufficiently firm and unjointed to stand without support.

In considering the strength or thickness of the arch, the following simple facts must be borne in mind: The weight of coal-measure strata may be roughly averaged at about 1 ton per 14 or 15 cubic feet, more or less. Taking this as the weight, it will be seen that 1 cubic foot weighs between 140 and 160 lbs. A square foot has 144 square inches; and if we suppose that a cubic foot is made up of 144 prisms, or pieces of stone, 1 foot long and 1 inch square at each end, and further assume that each of these pieces weighs 1 lb., we should have the weight of a cubic foot equal to 144 lbs. This is a very convenient figure for the purposes of calculation, and for that reason is frequently adopted; and, though it may not be strictly accurate, it is sufficiently near for the present purpose. If a piece of stone, 1 inch square and 1 foot high, weighs 1 lb., a piece of the same section, and 2 feet high, would weigh 2 lbs.; a piece 10 feet high will weigh 10 lbs., and 1000 feet high, 1000 lbs. Thus a column, 1 inch square at the base and 1000 feet high, weighs 1000 lbs., and presses with that weight on the ground below; and a column, with a base of 1 square foot, and 1000 feet high, would weigh 144,000 lbs., and so on. It follows that, at a depth of 1000 feet, the overlying rocks have a weight or pressure of 1000 lbs. upon every square inch below, and, at a depth of 2000 feet, of 2000 lbs., and so on. If, therefore, an arch at a depth of 1000 feet had to sustain the weight of the ground immediately over it, right up to the surface, it would probably be immediately crushed, as the following calculation will show. The width inside the arch is, say 20 feet, the side walls and arch, say 3 feet thick; total width, 26 feet, or 312 inches, therefore the total pressure upon every inch in length of the porch is 312,000 lbs., and this has to be borne by the two walls, each 36 inches. Dividing the 312,000 lbs. by 72, the pressure is found to be 4333 lbs. on every square inch of the brickwork. It is doubtful if any brickwork will sustain this pressure, which is twice the crushing pressure of common bricks, and equal to that of the best bricks, whilst many bricks will crush with one quarter of this pressure; but brickwork will not stand such a heavy pressure per square inch as single bricks.

It is, however, a well-known fact that a great many porches remain uncrushed for a generation, and sometimes at a depth greatly exceeding 1000 feet. It follows, therefore, that the weight of the overlying rocks does not rest upon the arch; in fact, it is only a comparatively small wedge of ground (see *n* Fig. 208, *b*) that is sustained by the arch. The arch, in fact, serves to keep every piece of rock tight and in its place.

It is, however, also a common experience that arches at a depth of 1000 feet are crushed, and this in cases where the brickwork is excellent, and has been designed and completed in a manner entirely first-class. Where this has happened, it is because the natural strata on each side of the arch are not strong enough to stand the weight above them without crushing or yielding; the pillars have therefore subsided, leaving the hard and incompressible brickwork to sustain the entire weight of the ground above it, for which task it is unfitted.

Construction of the Arch.—For the purpose of an underground arch, the hardest bricks are the best; sometimes fire-bricks are used to give the requisite strength, but generally well-burnt red bricks are used. The mortar is usually made of lime mixed with clean, sharp sand, or else with engine ashes, which often make a very good substitute for sand; sand mixed with earth or clay is not good for the purpose. Where there is much water, hydraulic lime is used. This lime is generally made from limestone of a particular kind, such as the blue Lias limestone, found at Barrow-on-Soar and in the district, also near Rugby; there is a hydraulic limestone found in some parts of Flintshire, near Holywell. Some of the magnesian limestone formation (Sutton) gives a hydraulic lime. The peculiar property of hydraulic lime is that it will set when wet, or even in water, and is not dissolved by the action of water. Instead of hydraulic lime, Portland or Roman cement is often used. In the neighbourhood of Bagillt, in Flintshire, a kind of limestone is quarried, which, when burnt and ground, makes a capital Roman cement. Roman cement is a quick-setting cement, setting in a few minutes; Portland cement is a comparatively slow-setting cement; the rapidity with which it sets can be regulated by the manufacturer, so that it may set either in half an hour or in a day or two, when used in building.

As the side walls are built up, all spaces between the wall and the natural ground are filled up with small stones and dirt or sand; and over the top of the arch the space is carefully rammed with small shale or with sand. The object of the careful ramming is to ensure uniformity of pressure all over the arch, so that it may keep its shape. Where ordinary brickwork will not stand without crushing, wooden bricks, or blocks, with open spaces alternating with each block, are substituted in layers of say 1 foot high, and at intervals of 2 or 3 feet; so that, if the natural strata should sink, the crown of the arch may also sink to a corresponding extent by the crushing of the wood. And it is said that success has attended this device. In some cases the arch is entirely made of timber blocks. In mines where, owing to the great depth, or the softness of some strata, arching will not resist the pressure, it is advisable to have as little as possible, if any. Timber or steel bars, supported on props, can be more easily and cheaply replaced when, owing to the pressure of the roof and sides, they have been pressed out of shape and position. It must be borne in mind that the function of all arching, propping, and barring is not to sustain the weight of the great mass of ground, but simply to keep small pieces of the roof from falling down, and so loosening larger pieces above. If the whole of the surface of the roof of a road or heading be supported by bars, over which are laid poles or planks, no pieces of ground can fall out.

The porches having been constructed, the levels are driven forward. Experience alone can show whether props and bars are necessary. The timbering of mines is dealt with in another chapter.

By whatever system a coal-mine is worked, it is the general rule—to which there are but very few exceptions—to leave a pillar of coal or other mineral near the shaft to protect it and the engine-houses and the machinery on the pit-bank from being injured by the falling down of the surface. The size of this pillar is not fixed by any invariable rule, but by the practice of the district. From measuring the pillars at many collieries in Great Britain, it would seem that the radius of a circle representing the area of the shaft-pillar should be about one-third of the depth of the shaft; that is to say, if the shaft were 300 yards deep, no coal should be got within 100 yards of the shaft on either side, or the pillar should be 200 yards in diameter. The pillar is generally set out

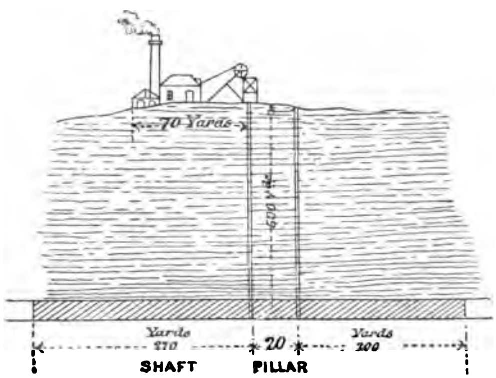


FIG. 209.—Shaft-pillar.

square, so that it should be 200 yards square. In the opinion of some mining engineers, this pillar is too small; according to the practice of others, it is, perhaps, larger than necessary. But if it is desired to maintain the shafts and buildings intact, without risk, it is certainly advisable not to make the pillar any smaller than that given in the rule. It is obvious, however, that with engine-houses and other buildings at some distance from the shaft, the shaft-pillars above set out will not be sufficient for their protection, and for absolute security it will be necessary to increase the width of the pillar. If these buildings reach a distance of 70 yards from the shaft, which is, say, 600 yards deep, it would be necessary to have the pillar on that side of the shaft extending to a distance of 200 + 70 = 270 yards from the shaft (see Fig. 209) Unless a sufficient pillar is left, it would be better to leave none at all, excavating the mineral, and filling the place completely with packs, so as to let all the strata above subside evenly.

In seams of considerable inclination the shaft-pillar, or any pillar left for the protection of surface buildings, should, according to some mining engineers, be larger on the rise side; the reason for this being that they assume that the line of fracture of the measures will be somewhere between a vertical line and a line at right angles to the dip of the strata. Fig. 209*a* shows a shaft-pillar in a seam inclined at 30°; the total diameter of the pillar may be the same as if the seam were level, but the distances to each edge from the shaft-centre are proportioned so as to give protection for an equal distance from the centre of the shaft at the surface. The evidence, however, in favour of this or any opposing theory is not so complete as might be desired.

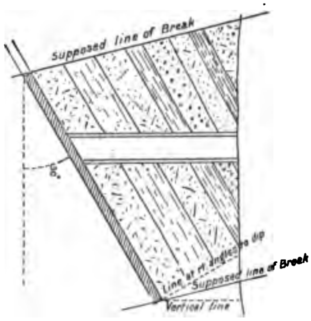


FIG. 209*a*.—Shaft-pillar in inclined seam.

In order to open out the mine, headings have to be driven through the shaft-pillar, and sometimes beyond, to open out the working places. Machinery has lately been introduced to facilitate the cutting of headings in coal. Stanley's coal-heading machine, referred to in another chapter, has been used for rapidly opening out a colliery. With the aid of these machines a heading can be advanced at the rate of 50 yards a week in hard coal, the machine making a heading 5 feet 4 inches in diameter, which

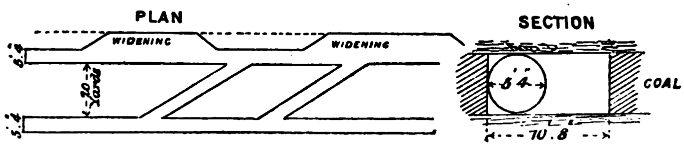


FIG. 210.—Heads driven by Stanley's heading-machine.

is afterwards widened by ordinary mining operations (see Fig. 210), two sets of men being employed in the widening operation, to keep pace with the machine. Great care is required in the use of these machines in fiery seams, because the ventilation of the heading depends on the compressed air by which they are driven, and is therefore intermittent. This might lead to accumulations of gas, unless precautions are taken.

In metalliferous mines, the opening out of a vein takes place at numerous openings from the shaft, say every 20 yards. The arrangements, therefore, at each landing are very simple, and the observations as to porches do not apply except in the case of very large mines having a vertical shaft, to which most of the mineral is brought to be raised to the surface.

CHAPTER VII.

METHODS OF WORKING: COAL, IRON, TIN, LEAD, COPPER, GOLD, SALT, SLATE; SPONTANEOUS COMBUSTION.

ONE method of working is quarrying, or open holes (see Fig. 211). Here a hole, or wide trench, is dug down to the seam to be wrought, which is then excavated and thrown into waggons, the strata above being thrown in a heap behind as the work advances, as shown in the figure. The soil is first of all removed, and is subsequently relaid on the waste-heap, so that the ground which has been mined can be restored to a state fit for agricultural purposes. This method of mining has been largely employed in most of the coal-fields of Great Britain, for seams of coal and ironstone where they crop up to the surface, and at depths not often exceeding

FIG. 211.—Open hole.

30 feet. At the present date, it is chiefly practised for the working of ironstone in Lincolnshire, Northamptonshire, Leicestershire, and other counties where Oolitic stone is found.

Fig. 212 is a photograph taken by the writer, showing an "open hole" at Mechernich (Rhine province of Germany), where lead ore has been got. In this case the whole of the ground excavated, from a few feet below the surface down to the bottom, contains ore. It is a rock of the New Red Sandstone formation, and the lead ore (galena) is distributed in specks, varying in size from a small pin's head up to a pea, and in some cases larger. The excavation shown in the plate is between 300 and 400 feet in depth, and 1000 feet in length. There have been also, in Great Britain, huge "open holes" for the excavation of copper ore, as, for instance, at the celebrated Parry's and Mona mines in Anglesea; also for the excavation of hæmatite, as in the Ulverston district, and in Glamorganshire, as at the Mwyndy and other mines. Large anthracite open mines have been worked in

FIG. 212.—Open hole at Mechernich.

Pennsylvania. In fact, this method of working is pursued wherever mineral is found at a small depth below the surface, because it is the cheapest.

Bell Pits.—An ancient method of working, practised within

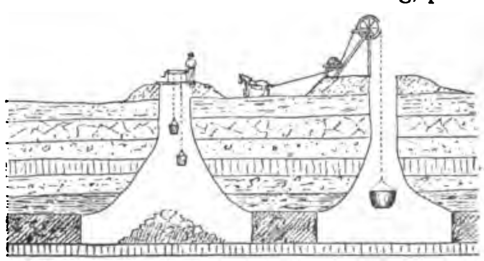


FIG. 213.—Bell pits. Ancient method of getting coal and ironstone, practised at the present time in some places.

the writer's recollection, is by bell pits (see Fig. 213). A small pit, sometimes only 4 feet in diameter, was sunk, and belled or widened out at the bottom to just such a diameter as might seem safe, having in view the risk of the undermined ground falling on the workmen. Several rows of ancient bell pits, sunk side by side along the outcrop of some seams of coal and ironstone, may be followed for miles in some coal-fields. It is evident that this method of working is only suitable for very shallow mines.

Pillar-and-Stall. — This is the most general system of working mines, whether coal, iron, clay, stone, slate, salt, gold,

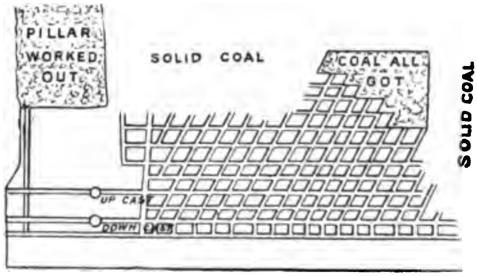


FIG. 214.—Plan of a pillar-and-stall coal-mine.

silver, or other mineral, although it goes by various names. When applied to coal-mines it has many aliases, such as *stoop-and-room* in Scotland, *post-and-bank*, etc. Figs. 214 and 215 show plans of pillar-and-stall mines. The stall is the place from which the coal or other mineral is excavated in the first instance; and the pillar, as its name implies, is the pillar of coal or other mineral which is left to support the superincumbent strata.

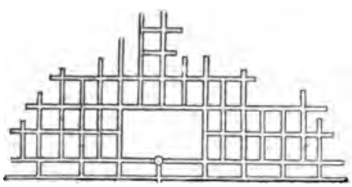


FIG. 215.—Stoops (18 yards × 12 yards) and rooms (4 yards wide).

The workmen cut the whole of the seam of coal into pillars and stalls. The width of the stalls depends upon the strength of the roof and other conditions; for instance, if the shale above the coal will make a sound roof to the stall—if it is 6 feet wide, and would fall down if it were 8 feet wide—that would be a good reason for having the stall only 6 feet wide. On the other hand, it is often more expensive to get the coal in a narrow stall than in a wide one, because each side of the stall (see Fig. 216) may have to be cut, either with the pick-axe or by the action of an explosive; and this cutting, when done, would suffice for a stall either 2 yards or 20 yards in width. In coal-mines the stalls are usually from 4 to 6 yards wide, and the roof, unless very strong, is supported by props, and sometimes by packs of dirt (see Fig. 217). The width of the intervening pillars also depends on a variety of circumstances. Thus if the pillar is very hard and strong, it will carry a heavier load than if it is soft, and therefore a narrow pillar will suffice. The pillars are sometimes left only half a yard in width between stalls 4 or 5 yards wide; but, even if the coal were very strong, such a narrow pillar would crush very soon in a mine exceeding 15 yards in depth. It is evident that the deeper the mine, the greater the weight the pillar has to carry, because, the whole of the mine being divided into pillars and stalls, there is no other support for the overlying rocks than the pillars of coal; and at a depth of say 1000 feet, the pressure being 1000 lbs. per square inch, that pressure has to be borne by every square inch of the upper surface of the seam of coal. If, therefore, half the coal is cut away, and the roof remains unbroken, the whole of the original weight has now to be borne by the remaining half, and therefore the pressure per square inch is 2000 lbs. Taking the case of a mine only 100 feet deep, the original pressure per square inch would be 100 lbs. If nine-tenths of the coal were cut away, the remaining tenth would have to bear the whole of the original pressure, equal to 1000 lbs. per square inch. It thus appears that, at the shallow depth of 100 feet, pillars equal to one-tenth of the original area have to bear a less pressure per square inch than the pillars equal to one-half the original area at a depth of 1000 feet. The pressure, however, that a pillar of coal will bear probably increases with the dimensions of the pillar; thus a small pillar would crush with a pressure of 1000 lbs. per square inch, while a large pillar would bear the same pressure without injury. But it is not only the fact of

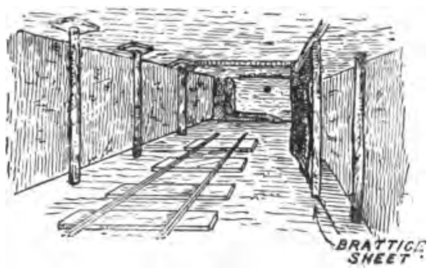


FIG. 216.—View of stall.

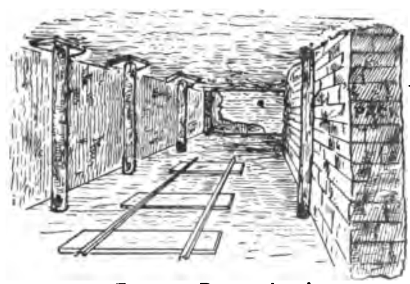


FIG. 217.—Props and pack.

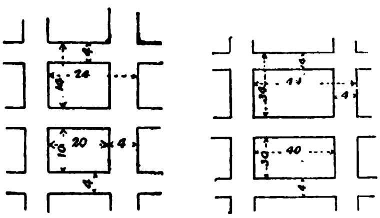
pressure on the seam of coal that has to be considered. The coal might be exceedingly tough and hard, and capable of resisting a pressure of 3000 or 4000 lbs. on the square inch, but the underlying seat-earth, or fire-clay, might be less strong, and if the pressure were to be great, the pillars would be forced down into this fire-clay until the roof rested upon it. The fire-clay would swell up. When this happens it is called "creep" (see Fig. 218). In most pillar-and-stall mines there is some creep, but not to such an extent as to cause the complete filling up of the stalls. Another effect of excessive pressure is upon the roof. If the shale constituting the roof is softer than the coal, or less tough, the roof may be forced down between the pillars which cut up into the roof. This cutting of the roof at the edge of the pillars takes place generally in all coal-mines, unless the roof is unusually hard, or is composed of a thick-bedded sandstone. The ordinary shales of the coal measures are thin-bedded, and although it would require a very great force to break through several feet of these shales at once, the lowest bed, perhaps only 1 inch thick, soon

FIG. 218.—Section of pillars and stalls.

gives way, and when that is broken, the next bed above has to bear the pressure, and so on, till by degrees a great thickness of roof is broken down. Thus it happens that a roof may be strong enough to stand good for a few days, but at the end of a fortnight a good deal might have fallen, and at the end of a month, the stall might be nearly filled with fragments of roof. In some cases the roof is sufficiently sound to remain for thirty years, and more, as it was when first exposed.

It is evident that a roof may be a very good roof, that is to say, it may stand firm in a mine that is only 100 feet deep, when exactly the same shale roof in a mine 1000 feet deep would be broken by the extra pressure, and in a mine 2000 feet deep would be still worse to deal with. In setting out, therefore, the size of the pillars, consideration has to be taken of the strength of the roof, coal, and floor, and also the market in which the coal has to be sold. For instance, if the roof and floor are very hard, the coal alone might suffer from great pressure, and the pillar sustaining this great pressure, though appearing whilst in the mine to retain its original form, might really be so crushed, that when it was cut with the pick-axe it would nearly all drop into small pieces, or slack. If, in the market where this coal was sold, a good price could only be got for large coal, it is evident that the effect of crush on the pillar would be commercially disastrous; if, on the other hand, all the coal were sent to the coke-oven, the size would be immaterial, and no loss would ensue from the crush. This consideration has a great deal to do with the method of working adopted in the various coal-fields of Great Britain.

Size of Pillars and Stalls.—The ordinary size of stalls has been above given as varying from 4 to 6 yards, and the ordinary size of pillars may be taken as varying from 5 yards wide and 10 yards in length to 30 yards wide and 40 yards in length. As the development of the coal-fields of the country takes place, the average depth increases, and the size of the pillars increases to a corresponding extent. To take an illustration, suppose a mine 1500 feet in depth, and the pillar 10 yards wide and 20 yards long, with stalls 4 yards wide, and cross-cuts the same width. The mine may be divided into rectangles (see Fig. 219) of 14 yards by 24 yards, equal in area to 336 square yards. Of each of these rectangles the pillar has an area of 200 square yards, and therefore this area of 200 square yards has to bear the pressure originally resting upon an area of 336 square yards, the original pressure being 1500 lbs. per square inch. The pressure per square inch upon the remaining pillar is found by the following rule-of-three sum :—



FIGS. 219, 220.—Diagrams of pillars, showing calculation of pressure.

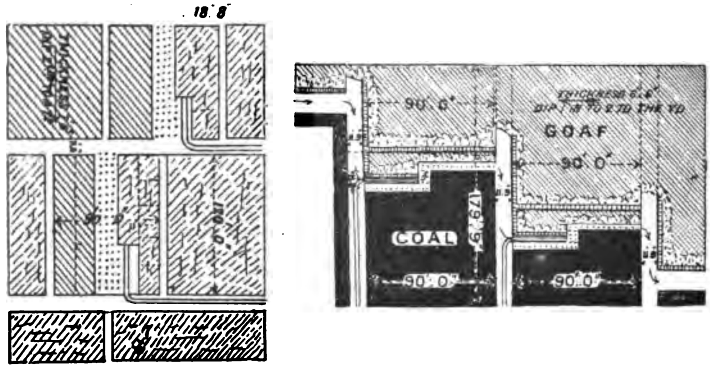
$$200 : 336 :: 1500 : x = 2520$$

It is probable that this pressure will be a great deal more than either the coal or the roof or floor will bear without injury, and the working of the mine will become exceedingly costly, if not impossible. Supposing the pillars are increased to 30 × 40 (see Fig. 220), the stalls remaining the same size, the whole of the mine may be divided into rectangles of 34 × 44 yards. Therefore each rectangle contains 1496 square yards, and each pillar contains 1200 square yards. The original pressure of 1500 lbs. per square inch upon the original area of 1496 square yards has now to be entirely sustained on the remaining pillar of 1200 square yards, and the pressure per square inch will be found by the following rule-of-three sum :—

$$1200 : 1496 : : 1500 : x = 1870 \text{ lbs. per square inch}$$

Thus, by increasing the size of the pillar, the pressure per square inch upon it has been reduced from 2520 to 1870. These are some of the considerations that guide the mining engineer in setting out the dimensions of pillars. There are, however, other considerations, such as the inclination of the seam. In a level seam, or one lying at a moderate inclination, there is no difficulty in conveying the coal along the stall, but if the seam lies at a steep inclination, special arrangements have to be made for haulage, which will be dealt with in the chapter on haulage. Other considerations are the amount of gas yielded by the coal, and the means of ventilation. It is evident that the larger the pillars, the greater will be the lengths of single road to be ventilated between cross-cuts; and, in a mine yielding a great deal of fire-damp, this is a very important consideration, of which notice will be taken further on.

Getting the Pillars.—A hundred years ago, the colliery owner was not often troubled by the consideration where he would find the coal, but generally he had only to consider where he would find the market, and it was not necessary for him to trouble his mind by asking himself if he intended to get the pillars, and how. Sometimes the miners would find it easier to get the coal out of the pillars left for the support of the stalls (which were in fact the roads necessary for the transport of the mineral, and for ventilation), than to cut coal out of the solid, and therefore they would make a stall across a pillar, or cut coal off the edges of a pillar, so weakening it that the roof would break down, or the floor heave up. This was called "robbing" the pillars, and it would seem to have been originally regarded as a somewhat nefarious operation. At the present date the working of the pillars constitutes from fifty to eighty per cent. of the entire working of the mine, and it is not put off until the whole of the coal-field to be worked by any given colliery has been cut into stalls, but is begun as soon as the workings have reached a sufficient distance from the shaft for this operation to be performed without danger to the colliery plant (see Fig. 214). The working of the pillars is often done in the way shown in Fig. 221, that is to say, by taking a stall along the edge of the pillar, say 5 yards wide. In order to protect the men against falling roof, rows of props are placed as shown in the figure, and sometimes a pack wall is built so as to form a sheltered roadway between the pillar and this wall (see Fig. 222).[1] When this stall has been worked across the pillar, the rails are taken up and some of the timber withdrawn, and another stall started by the side of the one just completed, until the whole of the pillar has been got.



FIGS. 221, 222.—Pillar working (Ryhope).

Another method (see Fig. 223) is to work the whole width of the stall at once, setting timbers along the face as shown, and as the stall advances, the timber is withdrawn, and the props which are not broken replaced close to the coal face where the men are working. After the props are withdrawn, the roof may fall, and of course will fall, as soon as a sufficient area is left entirely unsupported. In cases where the pillars are large, this is sometimes called "working back," and also it is sometimes called "longwall working back," but it is evidently only one variety of pillar-and-stall. Often, in order to increase the security of the miners, packs or stalls of stone—that is, shale, sandstone, fire-clay, or other material—are built at intervals along the working place (see Fig. 223). These reduce

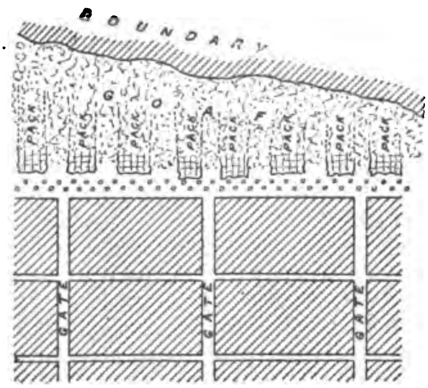


FIG. 223.—Working back.

[1] N.E Inst. French report.

the pressure on the timber, and remain as a permanent support for the roof; owing to their loose construction, they will not sustain a very great pressure, and they yield under the weight of the roof, and allow it gradually and equally to subside. The fracture of the roof is thus more regular than if there were no packs.

Shape of Stalls.—On looking at a plan, it will be seen that where the stalls cross there is likely to be a great pressure on the corners of the pillars. To reduce this, the stalls are often made narrow at the entrance (see Fig. 224), and gradually widened out to the full width.

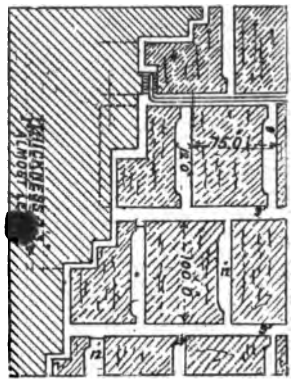


Fig. 224.—Stalls with narrow entrance.

"Bord" and "End."—Coal is traversed by lines of cleavage (see Fig. 225), along which it can be easily split by a wedge. These cleavage lines are close together; a wedge may be put in anywhere on the surface of the seam of coal, and it will split along the line of cleavage. These lines of cleavage are generally, so far as the writer has seen or heard, in a plane vertical, or nearly vertical, to the plane of stratification or bedding in the midland and northern counties. The direction of these cleavage planes in these counties is in line, roughly speaking, from north-west to south-east, but varying in the number of degrees west of north. In South Wales the cleavage planes are said to be generally not in a plane vertical to the bedding, but inclined at a considerable angle.

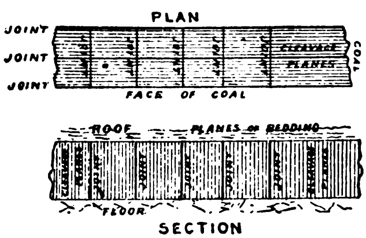


Fig. 225.—Cleavage planes and joints.

In addition to the cleavage planes are joints parallel to them, and others crossing them at right angles, the joints often make a complete separation, along which the coal can be parted; they sometimes contain black smutty coal powder. These joints are studied by the colliers, who make the holing or under-cutting so that the coal may be parted along a joint. The joints are not always regular nor completely cutting the seam, so that it is only sometimes that they assist the collier in his work. They are sketched with the cleavage planes in Fig. 225.

As a general rule, it is easier to cut the coal across these cleavage planes, just as it is easier to saw a log of wood across than to saw it lengthways. In taking a heading across the cleavage planes, it is called driving "bordways" or on the "bord;" in taking a heading in the direction of these cleavage planes, it is called driving "endways" or on the "end." In the north country the word "headways" is often used for "endways." When driving a heading in a diagonal direction, neither "end" nor "bord;" it is in Yorkshire called "andra." It being easier to cut the coal bordways, the pillars are generally made longer in that direction. Thus there are more bords than ends. In many places the endways stall is made narrower than the bordways, in order not to weaken the pillar unnecessarily. The

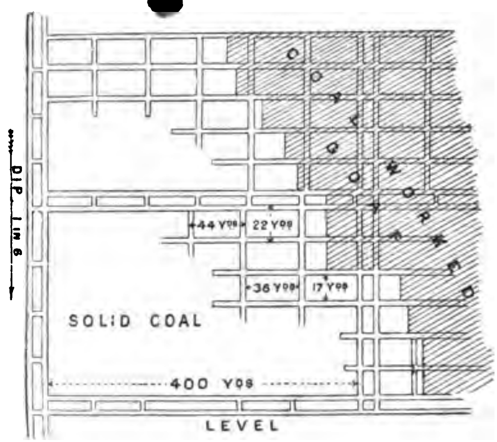


FIG. 226.—Cheshire pillar-and-stall. Coal worked shown thus //////.

nearer the cross-cuts or endways stalls are, the better for the purpose of ventilation, and a cross-cut 6 feet wide will be wide enough for that purpose. In places where the bordways stalls are 5 yards wide, the endways stalls are often only 3 yards wide.

Fig. 226 shows a method of working which has been adopted in some of the Cheshire collieries. The dimensions of the pillars are shown on the figure. In this case the roads or stalls are driven in pairs, with a narrow pillar between, so that cross-cuts may be frequent, and ventilation thus facilitated. A pair of roads can be driven out a great distance without difficulty, to the boundary; the next pair is about 140 yards distant, so that the mine is divided into pillars about 400 yards long, by 140 yards wide, these being so large, in proportion to the narrow width extracted in the headings, that the pressure is not perceptibly increased. The process of working back is begun at the boundary by dividing the large pillar into small pillars as shown in the figure, and then working back the pillars immediately they are formed, so that the stalls are only one or two pillars in advance of the "goaf" or "gob"—goaf or gob being the name given to the "waste" from which the coal has been extracted. By thus rapidly working pillars, there is not much time for the pressure to take effect either upon the coal, roof, or floor.

Fig. 227 shows the method adopted in North Staffordshire, where the coal has a steep inclination. The stone head or "crut" from the shaft foots the coal, along which levels are driven; from this level headings are driven uphill in pairs, as shown in figure. A heading driven uphill in North Staffordshire is called a "dip;" these dips are driven to the rise boundary, or pillar left against old workings above; levels are then driven through at the top. Half-way between the dips the pillar is cut in two, and each half worked back; as the top pillar is being worked back, a lower level is driven out. The levels are ventilated by air-pipes; the return air-road is driven in the seam of coal to the upcast shaft.

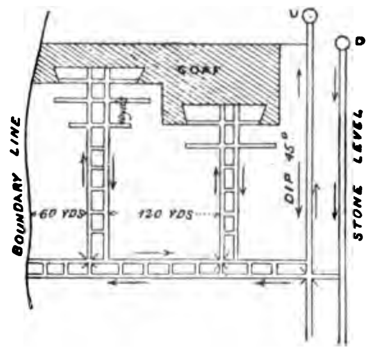


FIG. 227.—North Staffordshire pillar-and-stall.

Main Roads.—In working a large mineral estate, main roads may be driven north, south, east, and west. These main roads will go to the extreme boundary, possibly two or three miles distant. There must be at least two main roads in each direction, one for the intake, and the other for the return air. From these main roads branch roads start into the districts or panels of work, each side of the district or panel being worked in the manner already described. In getting the pillars in these districts, it is necessary to leave a sufficient number untouched near to the main road, in order that it should not be crushed. The width of solid coal which it may be necessary to leave on each side of a main road varies, of course, with the nature of the strata, as above described, and with the depth of the mine; the deeper the mine, the wider the pillar which it is necessary to leave. In a mine about 150 yards in depth, a main-road pillar about 88 yards wide will suffice. Out of this two main roads 3 or 4 yards wide may be taken, and cross-cuts. For a mine 250 yards deep, a pillar about 110 yards in width will suffice to protect a haulage road and air-road with cross-cuts. For a depth of 400 yards, a width of about 203 yards is required to protect a haulage road, air-road, and cross-cuts.

The system of working by pillar-and-stall is carried out in mines lying at any inclination, from those which are perfectly flat to those which are vertical.

"Longwall."—The other systems of working coal and other minerals may be taken under the head of "Longwall." In the case

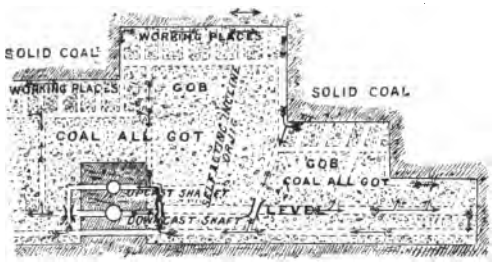


FIG. 228.—Longwall.

of coal-mines, there is only one invariable distinction between a longwall and a pillar-and-stall mine. In the pillar-and-stall mine pillars are left for the support of the roads for haulage and ventilation, and in the longwall mine no pillars of coal are left, but the roads for haulage and ventilation are supported by artificial

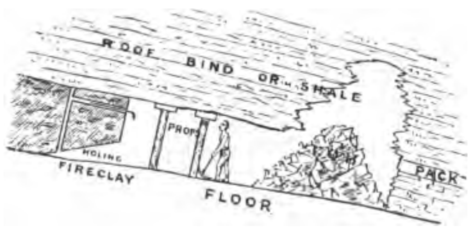


FIG. 229.—Section of a working place.

pillars or packs of stone or timber. The student, however, should bear in mind that in a great many mines there is some admixture of the two systems, and in many longwall mines—rightly so called, because they are chiefly worked on that system—the main roads are supported by pillars of coal. There are also, as above said, mines where all the roads are supported by pillars of coal, and where the coal is worked back in long working faces, which are often called "longwall working back." But these, as already said, are really pillar-and-stall mines. In the pure longwall mines the only pillars left are those for the support of the shaft or some other exceptional purpose. A plan of a longwall mine is shown in Fig. 228. A section of working place is shown in Fig. 229. Fig. 230 shows cross-section of gate road. Fig. 231 shows longitudinal section down the centre of gate road. The

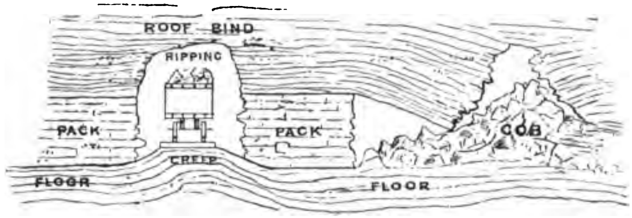


FIG. 230.—Cross-section of pack gate at *a b*.

place where the miners work is often called "face" of work. The working generally advances bordways; but where the coal is tender and it is desired to get large pieces, the face of work is carried endways, and very often it is taken "andra." In some collieries, where the coal lies level, or at a very moderate inclination, the workings proceed in every direction, so that the working face is a circle, of which the shaft is the centre.

As the coal is got, the roof is maintained by props, and when

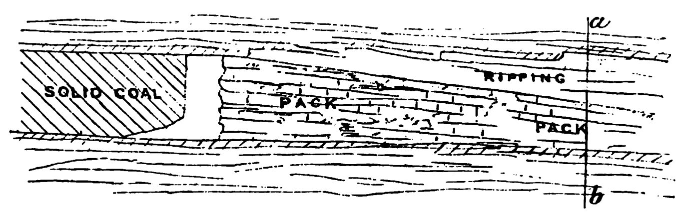


FIG. 231.—Longitudinal section of gate road.

the working has advanced a few yards from the shaft pillar, packs of stone are built where the gates will be. The gates are the haulage roads along which the coal is conveyed. The packs by the side of the gates generally vary in width from 9 to 12 feet. The interior of the pack is often filled up with small rubbish, and where the slack is unsalable some of it is often put in the packs. The width of the gate generally varies from 8 to 12 feet; the distance between the gates varies from about 10 yards to

100 yards (see Fig. 233, *a*). The nearer the gates are together, the more rapidly can the coal be got away from the face, and the greater the number of men that can be concentrated on a given length of face, and consequently the greater the tonnage of coal that can be got out of a given length of face. On the other hand, the further the gates are apart, the less will be the total cost of making and maintaining gate roads for a given acreage of the seam. As a general rule, the thicker the seam of coal the nearer the gates should be, to give facilities for removing the greater thickness of coal.

In some districts the miners form themselves into companies of a dozen or more to work the coal on each side of the gate road, and these twelve men may work a distance of 25 yards on each side of the gate road, being 4 yards apiece. In other districts the miners refuse to combine with each other, and each miner, who employs a labourer, has one side of the gate road to himself. In these districts the gate roads are placed from 10 to 20 yards apart, allowing each miner with his labourer from 5 to 10 yards of face. As the face of work advances, and the gates become longer, the packs become compressed by the weight of the roof, which no longer has the support of the solid coal, and, in order to maintain the height of the roads, it is necessary to cut down the roof. In the midland counties the term "ripping" is applied to this cutting down of the roof (see Fig. 231). Where the roof is hard, gunpowder or other explosive is generally used to break down the roof. The stone thus obtained is taken forward to the face of work, and used for building the packs. The gate roads in the midland counties, working seams from 3 to 6 feet in thickness, are generally from 30 to 50 yards apart. If, at the beginning, the gate road is made from 6 to 8 feet in height, it will probably remain high enough until the face is advanced a distance of 100 or 200 yards, after which it will require further ripping. In order to reduce the expense under this head, it is common to do the second ripping in only one gate road out of five or six, and to put a cross-road right and left from this main gate, so as to cut off the four or five roads on the other side, the hinder length of which is afterwards abandoned. This system of cross-gates not only reduces the cost of ripping, but also the cost of timber and road repairs generally, and the length of rails, number of sleepers, and cost of haulage. As the length of the main gates extends, main cross-gates are introduced, which cut off the branch gates of a whole district, and in large collieries the main gates and main cross-gates are worked by engine power (see Fig. 232).[1]

[1] T. Forster Brown, *Inst. Civil Engineers*, vol. lxiv.

Packing.—Between the gate-road packs it is a usual practice to build intermediate packs (see Fig. 233, *b*). These intermediate packs relieve the props from excessive pressure, and also, forming a permanent but compressible support for the roof, allow it to

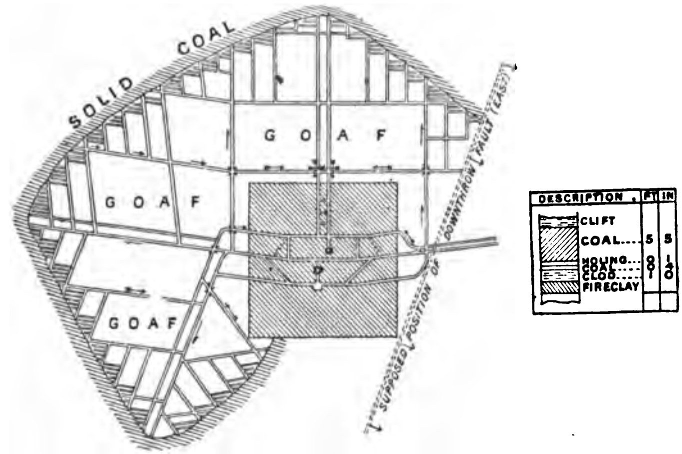


FIG. 232.—Longwall (Harris's Navigation).

subside gradually, and often without serious fracture of the strata above. The number and size of the packs that are built depend to a great extent on the building material at hand. In a thin seam of coal, where the holing is done in the floor, the coal being,

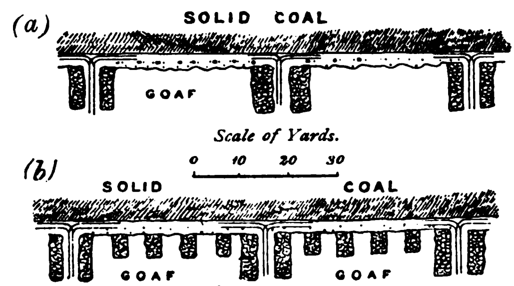


FIG. 233.—Plan showing packs in longwall face.

say, 2 feet thick, and the dirt got in holing 1 foot, the holing dirt will completely fill the goaf, and it will be unnecessary to build special packs, except by the gate walls. On the other hand, where there is a clean seam of coal 8 feet thick, and no dirt is got with the seam, material for building packs may have to be fetched from a distance. Perhaps the roof will break down behind the props, and from the heap of fallen stone so formed stone can be extracted for building packs. If, in first starting the working place from the opening off head, there is no packing material, and the roof is a strong shale or rock, it may be carried forward a distance of 30 or 40 yards or more, partly supported on timber props, before the roof breaks down. When it does fall it will be a very serious affair. The props being absolutely useless for the purpose of sustaining the weight above, safety can only be found in flight, and it is not unlikely that with the fall of the roof there may be an outburst of gas which has been pent up in the strata above, and, should there be a naked light in the vicinity, it might lead to an explosion. It is advisable to avoid these heavy falls of roof by bringing into a new district stone from other places, from the ripping of gates, or exploring heads, or cross-measure drifts. If sufficient stone cannot be obtained, the gate roads are formed by packs of timber, built in pillars from 3 to 4 feet square. These packs are made of bars of wood laid crosswise, forming a square pillar, the middle being filled up with slack or other rubbish. The cheapest kind of wood is generally used, often branches of trees of no use for carpentry; these are cut to the required length, and sold as "brattice" wood. Brattice wood is extensively used in South Wales, and in some parts of the midlands. As soon as the weight comes on these timber packs, they are crushed, and the height of the road can only be maintained by ripping down the roof. The stone thus obtained is used for forming packs, being built between the timber packs. In France it is usual, where thick seams of coal are worked, to quarry shale and stone on the surface, and send it down the pit for building packs. The extent to which the packs are squeezed down varies, of course, with the proportion of material in the pack and intervening spaces or "bays" into which they can spread when pressed, and also with the depth of the mine. In mines, however, 200 yards deep and upwards, the packs are generally so tightly crushed and forced into the underclay, which rises up between them, that they are completely buried, and the main roads are formed entirely in the roof of the coal by successive rippings. If this roof is formed of moderately sound shale, such as is generally met with, the road thus made entirely in it will endure, with a very small subsequent expense for repairs. The longwall system is becoming more and more general every year; for fifty years it has been the system almost exclusively adopted in Derbyshire and Nottinghamshire; it is now becoming the rule in Yorkshire, and is no longer uncommon in the northern coal-field. It is often practised in Scotland, and is common in

Lancashire, and is the rule in North and South Wales; it is also the rule in Leicestershire, in some parts of Warwickshire, and in the Cannock Chase district of Staffordshire. It is largely employed on the Continent.

Comparative Merits of Longwall and Pillar-and-Stall. —As a general rule, longwall is the system employed where it is essential to get coal in large lumps, and where there is packing material to build the walls or packs. With seams of moderate thickness, and where there is dirt to build the packs, the ventilation of the longwall mine is simpler than with a pillar-and-stall mine. In thick seams of coal, where in ordinary course of working there is little or no dirt for the building of packs, pillar-and-stall is generally employed. In driving out the stalls in a fiery seam, it is often difficult to keep them clear of gas, because these

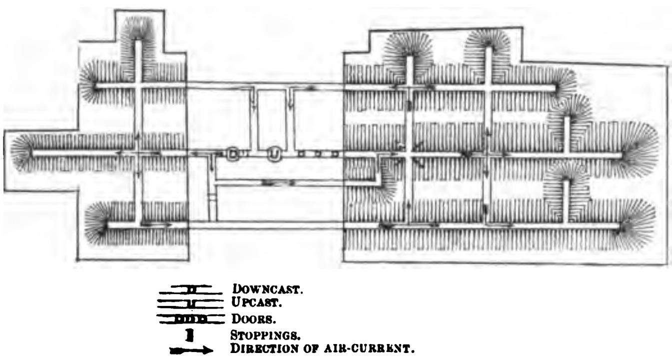


FIG. 234.—Drainage of gas by headings. Proportion drained, 10,000; proportion worked, 2000; ratio, 5 to 1. Drainage of gas shown by shading.

narrow roads receive, not only the gas due to their own area, but the drainage of gas from the adjoining pillars (see Fig. 234). In this case it is shown that the area drained of gas by the stalls or headings is five times as large as the area from which the coal is extracted. At a later stage in the working of the mine, when the pillars are got, the proportion of gas yielded per ton of coal will be much less; but this subsequent immunity may be dearly purchased if at the beginning the mine is fouled by the excessive drainage of gas. On the other hand, in a longwall mine the drainage is proportional to the tonnage of coal got at all stages in the life of the colliery, after the first headings have been made through the shaft pillar. The ventilation of a longwall mine is more satisfactory than that of a pillar-and-stall mine in the following important respect. All the roads, both for haulage and ventilation, lead through the goaf, and thus there is no large unventilated area in the mine. The coal being extracted, leaving no pillars, the superincumbent strata subside and close up entirely the opening made by the extraction of the coal, except where the gates have been made, and in the vicinity of the working face, where sufficient time has not elapsed since the coal was got to allow of the complete settlement of the rocks above. On the other hand, in a pillar-and-stall mine there are, as a general rule, no roads through the goaf, so that the area along the edge of the pillars which is not yet squeezed tight by the weight of strata above is unventilated and frequently contains gas. When the barometer falls the gas expands, and, unless there is good ventilation around the edge of the pillars, might reach to the working miners. A sudden and large fall of roof might drive the gas out of the goaf into the working places.

Spontaneous Combustion.—Wherever there are large heaps of small coal (slack), it frequently happens that the coal takes fire by a process of spontaneous combustion. It seems to be now accepted that this is due to the oxidization of the small coal, which produces heat; the heat is retained by the covering of slack on the outside of the heap, and the hotter the mine becomes the more rapid is the process of oxidization, until it proceeds at such a pace that the heat rises to the temperature of actual combustion, and extends from the inside to the outside of the heap of slack, where, meeting with greater quantities of air, combustion can take place freely, and the heap of coal then bursts into flame. Spontaneous combustion has been frequently observed on board ship where coal is stowed in masses of hundreds or thousands of tons, in heaps of small coal laid down in stock at collieries, as well as in the underground workings. In many mines a great part of the slack made in working is unsalable, and is therefore left. There are other portions of the seam of coal which are of inferior quality, and therefore unsalable, and there are also other portions of the seam which cannot be taken out because they are covered with falls of roof, or which it is dangerous to attempt to get for fear the roof should fall on the miners. In some mines coal is left in from all these four causes, and these mines are liable to spontaneous combustion.[1]

Wax Walls.—Danger from these fires is obviated in a variety of modes. In some places, where the seam of coal is worked on the longwall principle, the gates have been lined with walls about 9 inches or 1 foot thick made of lumps of clay, tempered in the clay-hole on the surface. These continuous walls of clay (see

[1] Some kinds of shale are liable to spontaneous ignition; instances of this may be found in Staffordshire, Derbyshire, Dorsetshire, and at Brora, Sutherlandshire.

Fig. 235) prevent the air from getting into the goaf between the gates, and in the absence of all air there can be no combustion. Where these walls are properly made, the only place where fire can occur is near the working face; this, however, is pushed forward rapidly, so that the heaps of slack are not exposed for long to any circulation of air. If, for any reason, the working should be stopped, such as through slackness of trade, strike, etc., a wax wall is taken along the face, so as to entirely exclude the air from the goaf. A bank of sand is sometimes substituted for clay.

This system has been extensively used in the Moira district of the Leicestershire Coal-field, but at present it is not much used. In working the main coal at Moira, the general practice is to head out and work back, the working face of several pillars being kept in a straight line; this is locally but illogically called "long-wall working back." Packs are built to support the roof of the working face; and if a long stoppage is expected, or signs of heating appear in the goaf, the packs are made into a continuous

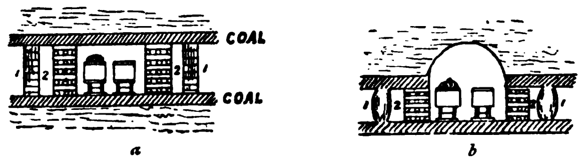


FIG. 235.—Gate roads resting on brattice wood and wax walls. *a*, before weight came on; *b*, after weight came on. 1, wax wall; 2, chocks.

wall and smeared with wax (see remarks on Spontaneous Combustion). In other districts where there is occasionally spontaneous combustion, nothing is done to interfere with the ventilation; but rather it is made as powerful as possible, on the theory that whilst a moderate supply of oxygen causes combustion, on the other hand, a powerful current so cools the place that the heat is dissipated before it has reached a dangerous point. It is probable that there is a difference in the nature of the coal which leads to a difference in treatment. If, however, a smouldering is discovered, it must be immediately dealt with. The plan generally adopted is to shovel out the heated or smouldering coal or shale, filling the place with sand. In order to get at the place, however, it is sometimes necessary to use water to extinguish the fire. The water for this purpose is sent in water-tight boxes or tanks on wheels, and to throw it on the fire a force-pump like a garden pump may be used. Occasionally the apparatus known as "extincteur" has been used. This apparatus contains an acid and some lime and water. By breaking the internal glass bottle containing the acid, carbonic acid gas is produced at a high pressure, and by opening the tap the water and carbonic acid gas can be projected to a considerable distance. It has, however, been found that water is sufficient. In some mines water-pipes are conveyed along all the main roads of the mine, and the water is brought down the shaft in pipes, so that there is a great pressure. By means of hose-pipes a powerful jet of water can be taken into any district of the mine to extinguish a fire. This is, perhaps, the best method of dealing with spontaneous combustion or fires of any kind. In South Staffordshire, owing to the great amount of coal and slack left in the workings, spontaneous combustion frequently occurs, and the mine is divided into panels, so that, when a fire takes place in any one of these panels, it may be separated from the rest of the mine by walls of brick and mortar called stoppings. In Warwickshire, a number of the more important collieries have their present workings to the dip of the shaft. To get the coal, headings have been driven down to the boundary, and the coal worked back. The water that finds its way down to the dip is left to fill the goaf, and thus prevents a fire.

As a general rule, a gob fire in Staffordshire or Warwickshire is "built off" to keep out the air, so that it may extinguish itself, the mine being so laid out as to facilitate the separation of parts from the rest.

Thick Coal.—The greater part of the coal-mines of Great Britain have seams of moderate thickness, that is to say, not

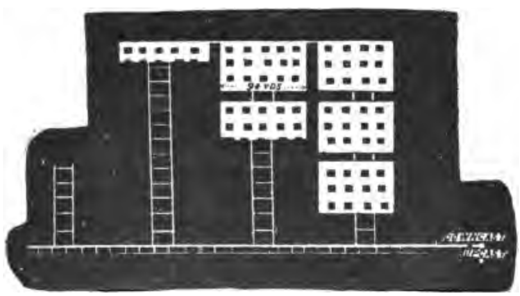


FIG. 236.—Method of working the Staffordshire thick coal.

exceeding 10 feet, and also lying at a moderate inclination. There are, however, districts with exceptionally thick seams, and others where the coal lies at a very steep inclination, and special methods of working are employed in each case. The thick coal districts in Great Britain are South Staffordshire and Warwickshire. The South Staffordshire thick coal is from 8 to 12 yards in thickness; the plan of a mine in this seam is shown in Figs. 236, 237, 238. The horse-roads are made in the bottom part of the coal, and the side of work headed out, the height of the stalls being from 6 to 8 feet, the pillars being left as shown. The roof coal is then cut down in beds according to the joints in the seam. The thick coal is really a number of seams separated by thin beds of dirt. In order to get the roof coal, it is necessary to cut a nick all round the sides of the panel and the pillars; this nick is cut by the miner with a pick-axe, standing on a scaffold or a

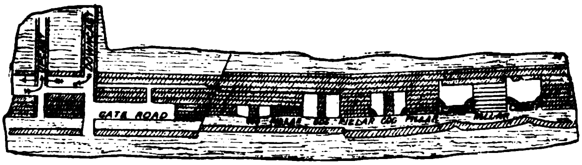


FIG. 237.—Section of rib-and-pillar system.

heap of dirt. In order that it may not fall on his head, "spurns" of coal are left at intervals of 2 or 3 yards. When a sufficient extent of roof coal has thus been nicked, the spurns are cut out by means of a long spear or pike, so that a man standing on the ground can do the work; little refuge holes are cut in the coal, where he can run for shelter as soon as the roof shows signs of dropping. In this way all the coal is got down except the top

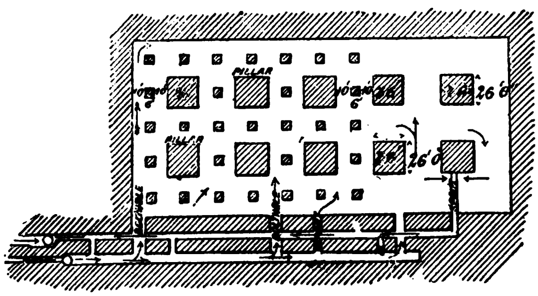


FIG. 238.—Plan of rib-and-pillar system.

bed, which is left, because when that is got down the shale above falls with it, and it is mixed with dirt. It would also be dangerous to enter the place to remove the coal, because more dirt might fall. The pillars and dividing ribs between the different districts, constituting a large proportion of the coal, are often left. Some times the pit has to be closed on account of the fires. After a time the mine is reopened, and workings are made in the pillars and ribs. A colliery may be closed and reopened three or four times before it is finished.

There are various modifications of the method of working thick coal above described. It is, however, impossible to work a thick seam like this without great waste, unless either long timber props are used to make the roof safe, as is done in Silesia, or packing stone is sent down from the surface, as is done in France. In Upper Silesia there is a seam of coal 8 yards thick, which is worked in the method shown in Fig. 239. The headings are made in the bottom of the seam. When the stalls are cut up to

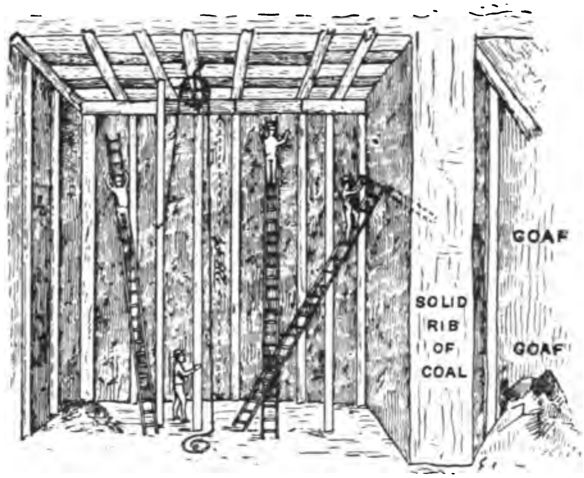


FIG. 239.—Method of working Silesian thick coal.

the roof, they are timbered with props and bars. No coal is left in the mine intentionally, and it is said by the managers that nine-tenths of the coal is extracted.

In the south of France, in the district of Grand Combe, there is a seam of coal about 21 feet thick, lying at a moderate inclination. This coal is worked longwall (see Fig. 240). In the first place, workings are set out in the lower portion of the seam, taking a thickness of about 7 feet. In the main road is a cross-gate to the right; this has ten branch gates, 10 yards distance apart from centre to centre. The packs are made with stone sent down a drop pit from a quarry on the surface. From the bottom of this pit into the working places the gradient is down-hill, so that the stone waggon runs without any power required for haulage. When it has been emptied of stone it is filled with coal, and then goes down the cross-gate into the haulage road, the gradient of which is towards the winding shaft, where it is raised

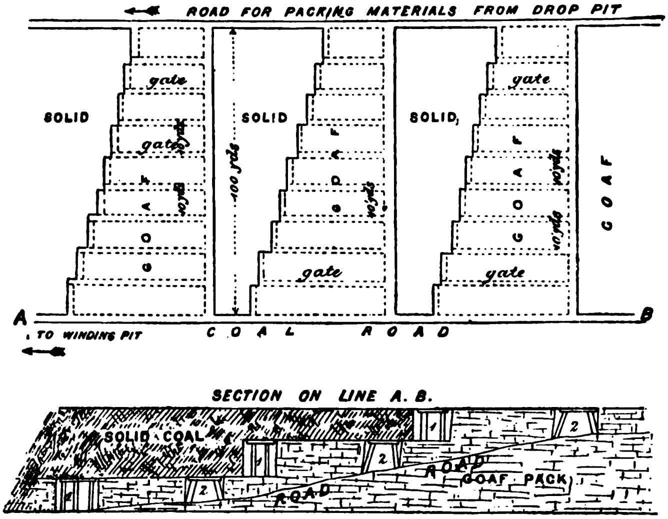


FIG. 240.—Grand Combe (France). 1, stalls; 2, cross-gates.

to the surface. The weight of the superincumbent strata consolidates the packs, and a new cross-gate is built at a distance of say 80 to 100 yards from the first one. At the same time that the pack is consolidated, the floor of the seam is heaved up, and the old roads become closed. The roof of the first cross-gate is now cut down, and it forms a road in a layer of coal 7 feet thick just above the old workings, and gates are started in this upper layer and carried forward above the old gob. In a similar way a third set of stalls is made in the top part of the seam, also 7 feet thick. In this way the

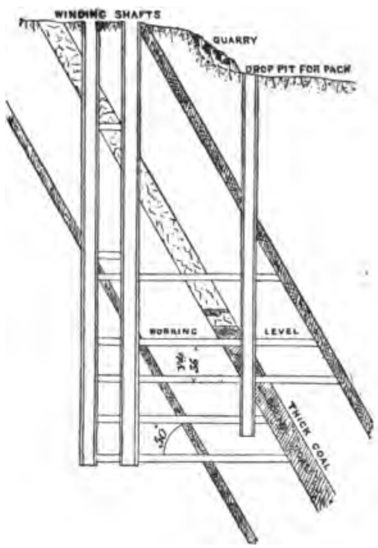


FIG. 241.—Cross-section of thick coal (St. Etienne).

whole of the 21 feet is got without waste, none of the coal is left in the pit, and the whole place is tightly stowed with stone from the surface. This method of working is very good; the cost

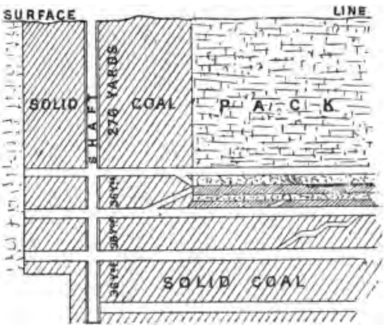


FIG. 242.—Longitudinal section of La Beraudier Colliery, St. Etienne.

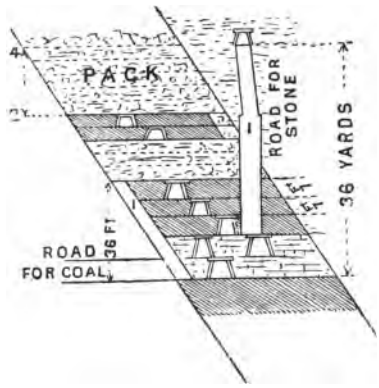


FIG. 243.—Cross-section of thick-coal workings (St. Etienne), enlarged scale. 1, inclined planes; hatched lines show solid coal. 2–4 was 11 metres, and is now squeezed into 8 metres.

of stowing a working place with stone obtained from the surface is said to be in some cases about 10*d*. a ton of coal got.

In the St. Etienne Coalfield, near the centre of France, there is a seam of coal about 60 feet thick, which lies at a steep inclination, about 50° from the horizontal. This coal is got by means of vertical shafts and cross-cuts (see Fig. 241). These stone cross-cuts are about 36 yards, one below the other.

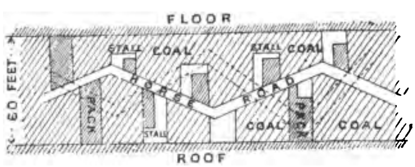


FIG. 244.—Horizontal plan. The roads are driven zigzag, so that those above are over solid coal, and not over roads below, except at the crossings.

From the cross-cut an incline is driven up in the coal, and a layer of coal about 7 feet thick is worked on the level across the seam from the roof to the floor; a central road is taken along the seam, and stalls started right and left, as shown in the plan. These stalls are stowed with dirt. When this layer is worked out, another layer of equal thickness is got below under the old gob, and so on until the whole 36 yards of thickness is worked out (see plan, Fig. 244, and sections, Figs. 242, 243). The stone is brought down a drop pit, and is lowered down a gentle gradient into the working places. The empty stone-waggon is then filled with coal, and again sent downhill to the winding shaft.

In Pennsylvania there is a splendid seam of anthracite, called the Mammoth Vein. The strata are much contorted, and the seam is in some places level, and in others vertical. The seam is worked by taking wide "breasts" up the hill, leaving pillars between, as shown in Figs. 245 and 246, cross-section and elevation. The seam here shown is 60 feet thick, lying at an angle of about 60° from the horizontal. The breasts of work are about 12 yards in width, leaving a pillar 10 yards between. By means of timbering on each side of the breast, a covered way is

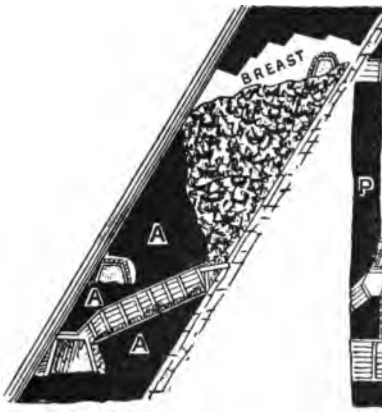


FIG. 245.—Double shute breast with batteries, steep dip: cross-section.

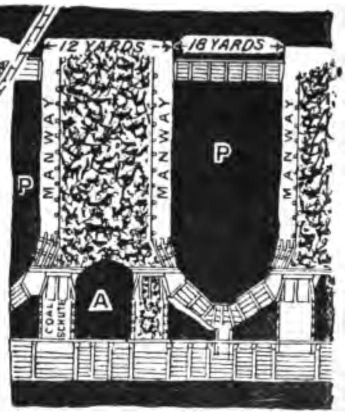


FIG. 246.—Ditto: elevation.

made for the workmen up and down. The coal is got down from the top, and falls in a heap under the workmen. A short heading leads from the bottom of the breast to the waggon-way; the coal falls down a slide laid in this heading into the waggons in the waggon-way. It can be stopped by means of a shutter; sometimes the coal at the face of the breast will come down without any working, and falls into the waggon by means of the shoot.

Vertical Coal-seams.—Coal-seams of moderate thickness, and lying at a very steep inclination, are worked in the method shown in the longitudinal section (see Fig. 247). Here the lowest level is timbered, and on the timbers another set of workmen can stand some yards back from the face of the first level; this place is again timbered, and other men standing on these timbers work the coal above, and so on. Stone from the sides is filled in on the timbers, so that the space between the gates is filled up with dirt. Shoots, down which the coal is thrown, are kept open by planks nailed to cross-bars. The bottom of the

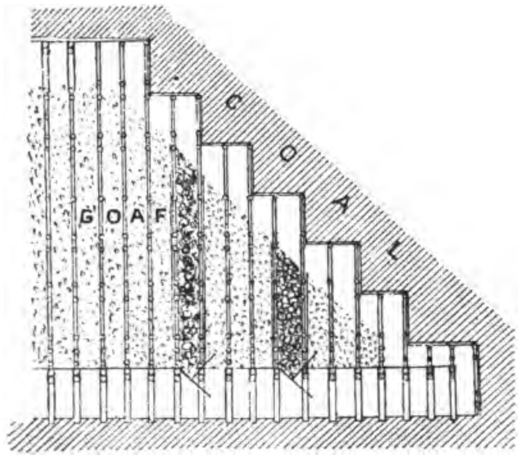


FIG. 247.—Method of working a thin vertical seam.

shoot can be closed by a wooden slide; when this is opened, the coal falls out into the waggon below.

Very Thin Seams of Coal.—In working very thin seams it is necessary to cut away some of the strata, either from the roof or from the floor, to make roadways of sufficient height; the cost of the roads is therefore increased. Sometimes the main roads are made a good height, say 6 feet, and large waggons pass along these; the branch roads are made no thicker than the seam, say 2 feet, and small boxes are filled with coal and drawn down the branch gate and emptied into the waggon (see Fig. 248). In many places these boxes slide along on runners like a sledge; in others, rails are laid, and the boxes have small wheels. In some mines small boxes on wheels go from the face to the winding shaft, and no large waggons are employed.

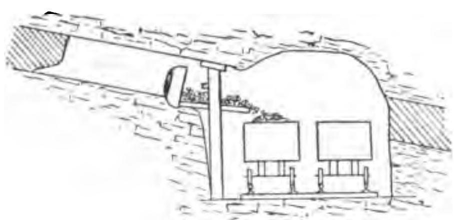


FIG. 248.—Section of gate road in thin seam.

Warwickshire Thick Coal.—In the southern part of the Warwickshire Coal-field six seams of coal come close together, making, with the dirt-beds which separate them, a total thickness of about 34 feet, of which, however, only 20 feet is got. They lie at a moderate inclination, as shown in Fig. 249. These seams are worked longwall, the workings in the lowest beds being in advance, and the other faces of work following. These workings are liable to spontaneous combustion. Most of the collieries are

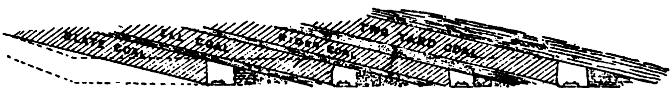


FIG. 249.—Section through working places in four contiguous seams (Warwickshire Coal-field).

now getting "dip" coal in the following manner. A pair of headings are driven down to the dip boundary, and then opened out on each side to a distance of perhaps 100 or 200 yards. Workings are then brought back as in a longwall face, beginning in the bottom seam, building the dirt into packs behind, and maintaining a road along the face to fetch the coal from the further end. Behind this face a similar one is opened out in the seam above; and, 20 yards behind that, another face in the third seam; and, 20 yards further back, a fourth face in the top seam. The air is brought down one of the headings, is turned to the left along two of the leading faces, goes down an air-road connecting the working faces at the side, returns along the lower face to the other side, then back along the two upper faces to the return air-road. The dip workings are allowed to fill with water.

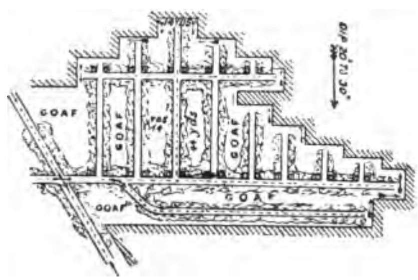


FIG. 250.—Great vein (Bristol Coal-field).

Bristol Coal-field.—In the Bristol Coal-field a method of working is adopted shown in Fig. 250.[1] This represents a level started out of a dip incline. The stalls are 14 yards wide; gate roads are packed with dirt. The thickness of the coal is about 3 feet. At the corner of each gate road a 4-feet-square pack of timber is placed. The coal is brought down the gates in sledges, and tipped into a waggon on the level. From the cross-gate to the lower level the roof is ripped for a jig road.

Varieties of Working.—In many cases the longwall face is

[1] Paper by Dr. Saise, North of England Institute.

not carried in a straight line, but one stall is in advance of the next, so that the line of face is in steps, as shown in Fig. 251. In this case the seam of coal is 5 feet thick, lying nearly level. The face is supported by props and by timber chocks about 2 feet square; the dirt is built into packs behind the timber and along the side of the gate and air-way; each stall is about 30 yards.

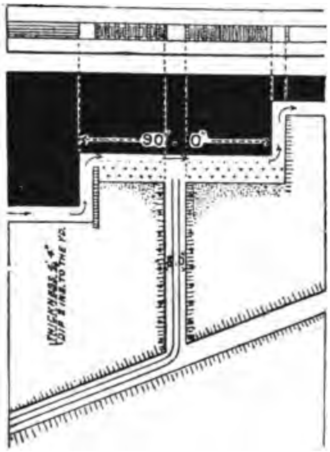


FIG. 251.—Yorkshire Colliery: longwall face in steps.

In coal-mines where the seam has a considerable inclination, as in Fig. 252 (which shows in sectional elevation a longwall working place at a colliery near Liege; Figs. 253 and 254 show some kinds of maundrils used in Belgium), the coal, after it is cut, slides downhill to the waggon-way, where it is loaded into the waggon.

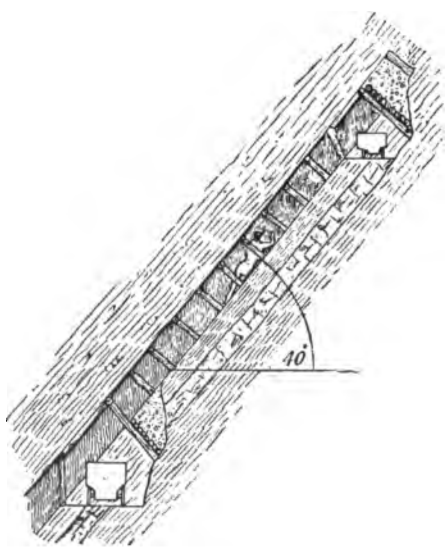
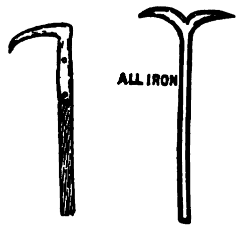


FIG. 252.—Section across level gates, showing longwall working place in steep measures.

In some Pennsylvanian mines headings are driven out in the coal, leaving pillars 15 or 20 yards wide. The pillars are then worked back, the roof breaking down behind; the dip being rather steep, the coal is sent down a sheet-iron tube called a "schute."

In Bohemia a seam of brown coal is got 30 or 40 feet in thickness, lying at a moderate inclination. Levels are driven out in the bottom of the seam, leaving a pillar about 25 yards wide; this pillar is worked by driving a stall across it about 7 yards wide, leaving a rib next the goaf 4 yards wide. This stall is only taken in the lower part of the seam, the coal above being supported by props. In working back, the props are sometimes removed in a rather interesting manner. Dynamite cartridges are tied to the top of some of the props, and discharged by a fuse. The props are thus knocked out, and the shock also shakes the coal so that it falls down in a great heap, giving the fillers sufficient occupation for several days.



FIGS. 253, 254.—Maundrils used in Belgium.

In many districts a modified system of pillar-and-stall, known as wide-banks, is adopted; this is shown in Fig. 255. In this case the banks are about 20 yards wide, leaving pillars of about the same width. Cross-cuts are cut through for ventilation about every 20 to 40 yards; a pack is built along each side of the stall, as shown in the section in the figure, forming a waggon-way; the roof is allowed to break down between the packs. The pillars

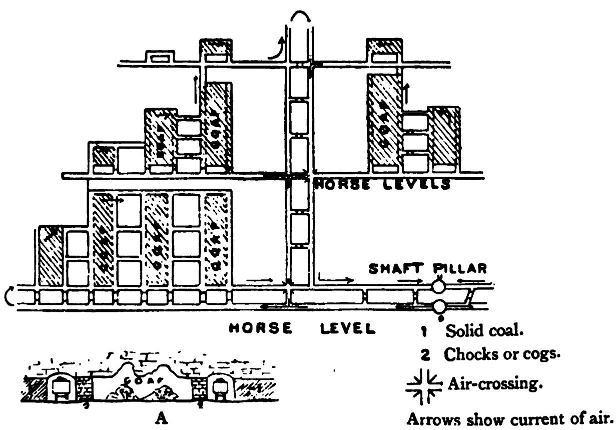


FIG. 255.—Pillar-and-stall, South Wales. A, section of double stall.

are afterwards "brought back" by means of the same waggon-way. This system is adopted in seams of various thicknesses from 4 up to 8 or 9 feet.

Metalliferous Mines.—The working of metalliferous mines may be understood by reference to Fig. 256, which shows a vertical section of a copper-mine, such vertical section corresponding to the plan of a level mine. It is seen that the mineral is got by vertical or nearly vertical roads called shafts, and by levels and by intermediate vertical shafts or cross-cuts called winzes. Workings are also made upwards from the levels called stopes, as shown shaded on the section. Pillars are left near the shafts

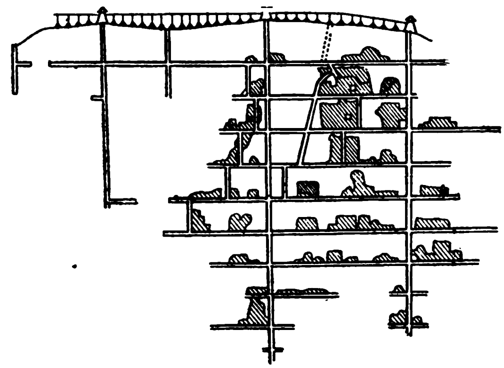


FIG. 256.—Longitudinal section of copper-mine.

to prevent them from being crushed. These stopes are made in the ore which the mine-manager considers most profitable.

In some cases the whole of the vein is got, in others only a small portion. A stope is shown on a larger scale in the vertical section Fig. 257, which is sketched from a gold-mine in North Wales. The mineral is got by blasting; the workmen above the bottom level stand on timbers wedged across the vein, over which planks are laid, so as to protect the workmen below. Sometimes the stopes are made downwards from the level, as shown in the vertical section Fig. 258; this is taken from a Cornish tin-mine. In this case the levels are 60 feet, or 10 fathoms, apart; the vein is here about 20 feet thick, of exceedingly hard rock, through which the tin ore is disseminated in minute particles. Compressed air power drills are used for heading; hand-hammer drills are used in the stopes. The

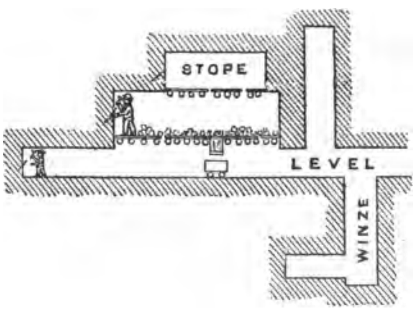


FIG. 257.—Gold-mine: longitudinal section of workings.

ore is thrown down a shoot into the waggon, which is run to the shaft, and there tipped into a hopper. It is drawn up the shaft in a hutch, which is filled from the hopper by drawing a slide. Cross-cut drifts are driven from one vein to another, as

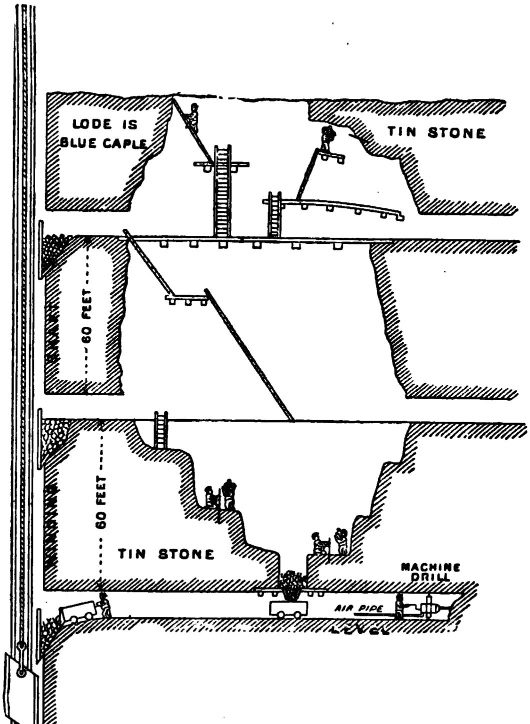


FIG. 258.—Method of mining tin near Redruth.

in Fig. 259, which is a section showing the veins and works of a gold-mine; a similar plan is adopted in most metalliferous mines.

Salt-mines.—Rock-salt is extensively mined in various countries; in England it is got at Northwich, in Cheshire, on the pillar-and-stall system, as shown in section Fig. 35, and in plan Fig. 260. The salt here lies in two beds, each about 25 yards thick. In the mine shown in the figures the lower bed only now is at work, and of this only the lower 7 yards are being mined. The pillars are left

12 yards square in rows 25 yards apart; the depth to the bottom of the mine is 110 yards. Although the pillars only constitute

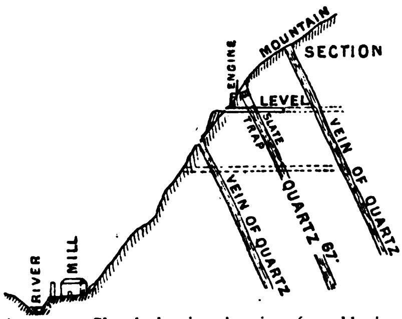


FIG. 259.—Sketch showing situation of a gold-mine in North Wales.

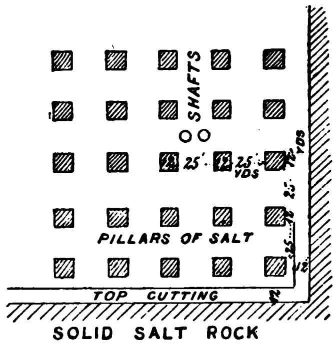


FIG. 260.—Northwich Salt-mine: plan of a part.

about one-tenth of the area of the mine, and have a pressure of nearly 3000 lbs. on the square inch, they show no signs of crush-

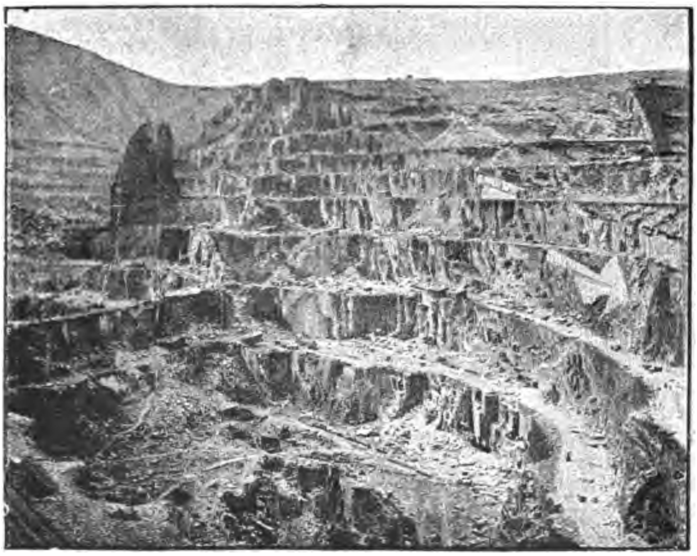


FIG. 261.—Penrhyn Slate-quarry.

ing. A coal-cutting machine, driven by compressed air, is used for holing on the top cutting. After holing, the salt rock is got down by blasting, and the lower part of the seam is got up by blasting. The shafts are tubbed through the boulder sand to keep out the water.

Slate Quarries and Mines.—Slate-quarrying is a very important industry in North Wales. At the celebrated Penrhyn Slate-quarry there is an open hole, a photograph of which is shown in Fig. 261. This hole is worked to a depth of 900 feet from the upper workings down to the lowest workings. There are fourteen or fifteen terraces, each 60 feet high, and from 10 to 20 yards wide. There is a railway on each terrace, and branches to every place where the men are at work. The slate is got by blasting, gunpowder being used, except in hard places, where gelatine is used; the shots break the slate off along invisible lines of fracture

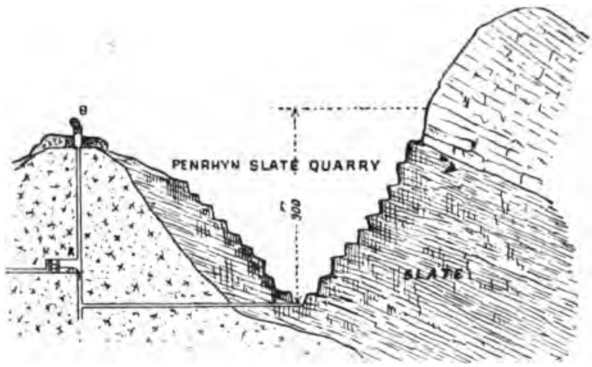


FIG. 262.—Section of quarry and shafts A, Hydraulic pumping-engine; B, winding machinery.

called pillaring lines, which are at right angles to the planes of cleavage. Every terrace is connected by a tunnel with a winding shaft. The mine (quarry) is in several divisions, so that the whole cannot be seen at a glance. One thousand seven hundred men and boys are employed at the quarry; every hour a horn is blown, when the men go into refuge-places, and the shots are fired. The beds are inclined at an angle of say 30° or 40° from the horizontal. The lower workings are drained by two hydraulic pumping-engines, the water-pressure being about 70 lbs. Each engine has three cylinders and three bucket-pumps. The buckets are 14 inches in diameter, with a 4-feet stroke; they deliver the water into a level which is 50 yards above the bottom of the quarry. The shaft is sunk into a rock locally called "granite," which is below the slate-beds. There are enormous heaps of waste slate, only about one-tenth of the entire production being made into marketable slates. Fig. 262 shows a section of the quarry and shafts.

In the neighbourhood of Ffestiniog, the slate is extensively got by underground workings reached by levels driven under the mountains, the workings being sometimes 1000 feet below the surface.

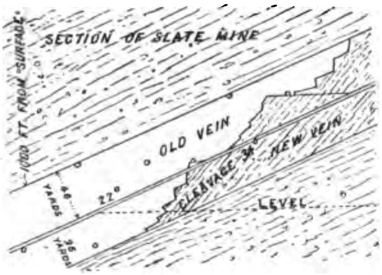


FIG. 263.—Section of working place.

Fig. 263 is a section of a working place. This shows two veins: the old vein, 46 yards thick, has a roof of chert, a hard siliceous rock; and the new vein below, 26 yards thick, separated by a bed of chert 2 yards thick. The beds lie at an inclination of 22° from the horizontal. The cleavage planes are at an inclination of 34° from the horizontal. The mode of working is pillar-and-stall; the stalls are called breasts, and are 45 feet wide, the pillars 24 feet wide; the levels are about 55 feet vertical distances apart. The slate is got by blasting; the blast-hole is made by a $\frac{7}{8}$-inch drill. The hole is *filled* with powder rather smaller-grained than ordinary blasting-powder. Very little tamping is done on the top, or else the slate would be shattered. The position of the hole is shown in Fig. 264. The shot, *a*, breaks off the slate along the pillaring line, *c d*. One end of the piece of slate is already loose from the previous blast, and the other breaks off along the natural joint. The bottom of the block separates from the mass along one of the planes of cleavage, as shown in the section. It will be seen from this figure that slate (and indeed every other mineral) has to be separated from the mass along lines in three planes. These three planes, in the case of slate, are, the natural joint, the pillaring line, and the cleavage line; and the separation of the lumps from the mass is facilitated by the previous operation having set free one end and the upper surface.

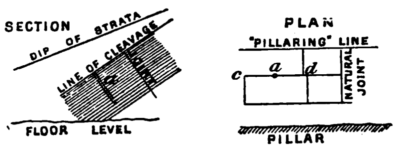


FIG. 264.—Slate-mine: blast-hole.

Hydraulic Mining.—Gold is often found disseminated in the diluvial or alluvial deposit of old river-beds. In order to reduce the labour of excavating the material and washing it, water is brought in wrought-iron or steel pipes from some high level, and is then directed through a nozzle upon the bank (see Fig. 265, which is a photograph taken by the writer in Colorado). The jet of water issues with such force that it knocks down fragments from the bank, which are carried away in the stream and down wooden troughs, where the gold is separated from the dirt. It is said that

FIG. 265.—Hydraulic gold-mining.

this process can be profitably applied if the amount of gold in 1 cubic yard of bank is worth 5*d*. This is a remarkable instance of economy of working, because, whilst the rate of wages in the Western States is a great deal higher than in Europe, 5*d*. would be a very low price even here for merely shifting a cubic yard of material.

CHAPTER VIII.

VENTILATION: DOORS, OVERCASTS, BRATTICE, PIPES, FURNACES, FANS: THEORY.

BOTH by law and by circumstance, it is necessary to produce sufficient ventilation in the mine to dilute and render harmless the noxious gases. It is, however, easy to produce a current of air in any place which has an inlet and an outlet, as in a dwelling-room, where an open window may serve for the inlet, and a chimney for the outlet. For the ventilation of mines it is necessary to have a similar inlet and outlet; it may be made either by levels, inclines, or vertical shafts. The same principle is observed throughout the whole mine, which is divided into inlets, technically called "intakes," and outlets, technically called "returns." These intakes and returns are separated by a pillar of the mineral worked, or of the strata in the case of cross-measure drifts, or by packs. In order that the air may pass from the intake to the return, it is necessary to make a cross-road connecting them, and the circuit of ventilation is then complete so far as this cross-cut.

Brattice.—As for that part of the main road which continues beyond the cross-cut, there is no thorough current of air; it is, therefore, necessary to divide this road (such road being locally called a heading, drift, bord, end, stall, etc.) by some artificial partition technically called brattice (see Figs. 266 to 270). This may be made by erecting wooden posts, either in the centre or a little nearer to one side, down the length of the road, and nailing from post to post boards, say $\frac{5}{8}$ inch thick. This brattice is connected with the side of the road before the cross-cut is reached; the air must now pass up one side of the brattice to the end of the road, and then down the other side, and so through the cross-cut, and then up inside the brattice of the second road, returning on the outside. Cloth is now generally substituted for wood (Fig. 267); this cloth is frequently a very coarse canvas, with some tar to save it from decaying; oilcloth is often used, which is less pervious to air. When cloth is used, a wooden lath is nailed against the roof from post to post, the cloth being nailed to this lath, any cavities above the roof being stuffed with bits of cloth or other soft material; the lower end of the cloth is buried in slack or dirt. If this brattice is carefully fixed against a smooth roof, and the cloth nailed tight to a post at every joint, and has a good air-space, say not less than 3 feet on each side, it will carry

Fig. 266.

Fig. 267.

FIGS. 266, 267.—Methods of bratticing.

a good ventilating current up to the face of the road or heading for a short distance. It will, however, only be by the exercise of the utmost care in fitting the joints of the cloth nailed at a post, that a good current can be carried for a distance of 50 yards. In the case of some very fiery headings, it is difficult to take a sufficient quantity of air even that distance; it is, therefore, necessary to put in the cross-cuts more frequently, say every 20 or 30 yards. In the case of some thick seams of coal, a brick brattice is used (Fig. 268); here the brick wall is turned in the shape of the quadrant of a circle over one corner of the heading, the cross-cuts being at distances of say from 50 to 100 yards, the bricks being pulled down and rebuilt. When it is desired to take a heading a long distance without a cross-cut, a brick wall is built from floor to roof, forming practically two roads (Fig. 269). If the top of the drift is too high and rough, the wall may be carried to a height of 5 or 6 feet, and boards placed across from this wall to the side of the heading, supported on a longitudinal bearer; the top of the boards is then covered with mortar, and the joints with the sides of the heading made with mortar (see Fig. 270).

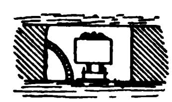


FIG. 268.—Bratticed brick quadrant.

Air-pipes.—Instead of brattice air-pipes are often used (Fig. 271); they may be made of wood of any required dimensions, say from 12 inches square inside up to 4 feet by 2 feet. A wooden socket is formed at one end of each pipe, into which the other pipe fixes, made tight with packing. Circular pipes of cloth are often used; light rings of wood or iron distend the cloth; this pipe is suspended by strings or wires from the roof. A cloth pipe is, of course, more leaky than a well-made wooden

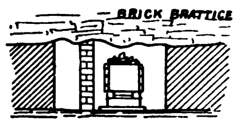


FIG. 269.—Bratticed brick wall.

FIG. 270.—Bratticed brick wall, covered.

pipe, and will, therefore, not take the current so far. Iron pipes are perhaps more generally employed than any other kind; these are made of sheet iron, with a longitudinal riveted seam, the iron being sufficiently thick—say $\frac{1}{32}$ inch, or more—to

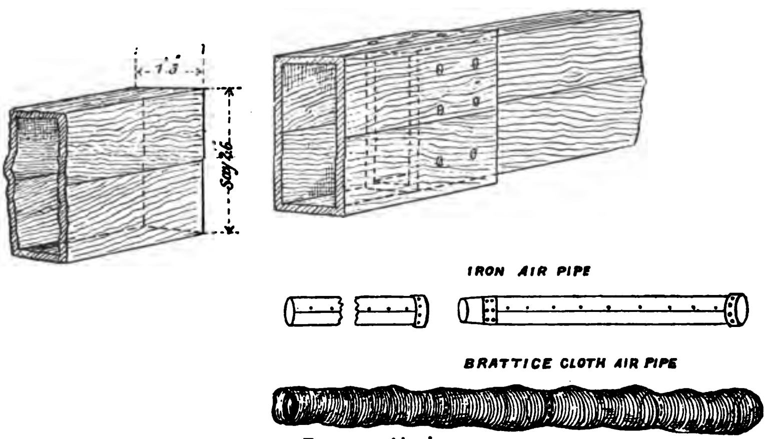


FIG. 271.—Air-pipes.

keep its shape. One end of each pipe is slightly tapered and fits into the larger end of the next pipe; it should be tight without packing, but the joint may be covered with a large flat indiarubber ring outside. These pipes may be laid on the side of the road, or suspended by wires from the bars.

Care required in using Air-pipes.—Pipes are often more convenient than brattice, but they must be used with great discretion if the ventilation is to be efficient. Take the case of a heading divided by brattice. The road is 6 feet high and 10 feet wide; the brattice near the centre has a return air-way of 6 × 4 = 24 square feet. Suppose, now, an air-pipe 14 inches internal diameter is used; it has an area of only 1 square foot, or $\frac{1}{24}$ that of the brattice. If, therefore, the pipe is so arranged that the intake air is blown through the pipe to the face of the heading, the small current so delivered, reaching into the face and to the top, may clear away the gas so that little can be detected upon examination. But all along the sides of the heading gas may exude, and 10 yards back from the face there may be an explosive atmosphere (see Fig. 272); this could not happen with brattice, because, owing to the leakage of air through, over, and under the

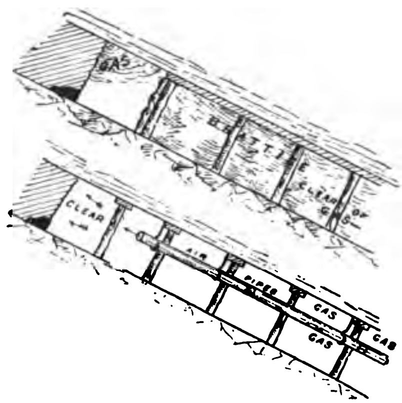


FIG. 272.—Application of brattice and air-pipes.

cloth, the face of the heading is always the part least likely to be ventilated, and if that is clear all else will be safe (see Fig. 272). The danger caused by a small pipe may to some extent be remedied by using a large pipe, say 20 inches in diameter; and not merely one line of pipes, but two, three, or even four, laid one above the other.

Method of fixing Pipes and Brattices.—Fig. 273 shows some bratticed levels in a seam of moderate inclination; the arrangement of brattice indicated is bad. Fig. 274 shows an improvement; it will be noted that in this case the lowest level is in advance of the second one, and that is in advance of the third one. The cross-cuts are so set out that they arrive at the place of junction just as the level reaches the same place; in this way the length of brattice is reduced to a minimum. In the second arrangement there is less hindrance to the air, and consequently a greater volume is conveyed to the end of headings. Fig. 275 shows an arrangement of air-pipes which is bad; it will be observed that the air returns from the face of the lower level through a pipe which passes up the cross-cut to the face of the level above, where it delivers the air which has been already fouled with gas from the lower level. Fig. 275*a* shows an

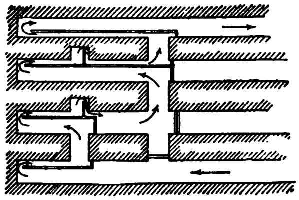


FIG. 273.—Brattice, bad.
Length = 72; Q = 2200; say 1600 will reach end.

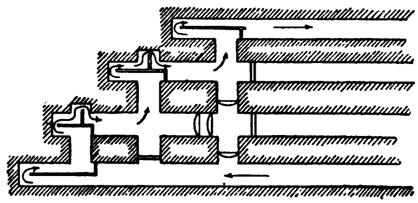


FIG. 274.—Brattice, good.
Length = 36; Q = 3200; say 3200 will reach end.

improved arrangement. Here the air-pipe from the lower level delivers the air through a cloth or other stopping at the bottom of the cross-cut; the supply for the upper level is taken by another pipe from the intake side of the stopping, and delivered into the face of the upper level; thus each level has a supply of fresh air. The length of pipe employed in each case is the same, but the resistance to the air-current is less in the second case, because

Fig. 275.

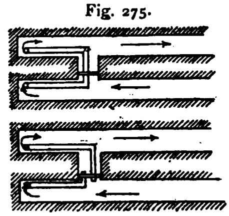


Fig. 275*a*.

Fig. 276.

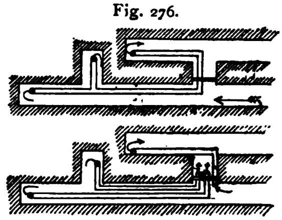


Fig. 276*a*.

FIG. 275.—Air-pipes, bad; Q = 260.
FIG. 275*a*.—Ditto, good; Q = 750.

FIG. 276.—Air-pipes, bad; Q = 200.
FIG. 276*a*.—Ditto, good; Q = 900.

there are two pipes instead of one. Fig. 276 shows another arrangement of air-pipes which is bad, and Fig. 276*a* shows a rearrangement of the air-pipes which is good. Fig. 277 shows a case where a number of stalls are started uphill from a pair of levels; if all these stalls were ventilated by brattice, it might be done in the way shown. Here each stall is divided with a brattice which crosses the upper level, so that the level is obstructed with a brattice for every stall that is started; on the other hand, if the stalls are, say, 12 feet wide, there is, allowing for the thickness of the props, an air-way 5 feet wide all round the brattice, and there is no reason why there should not be sufficient ventilation. Suppose the same work had to be done by air-pipes

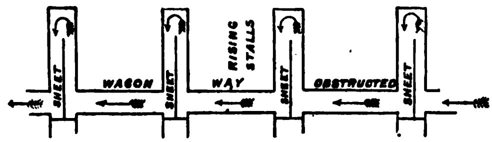


FIG. 277.—Brattice for a number of stalls.

which were to blow into the face of heading; the lower level must then be the intake road (see Fig. 278), and there must be a cross-cut opposite each stall. Each cross-cut is closed with stopping, and from this stopping one or more air-pipes carried across the

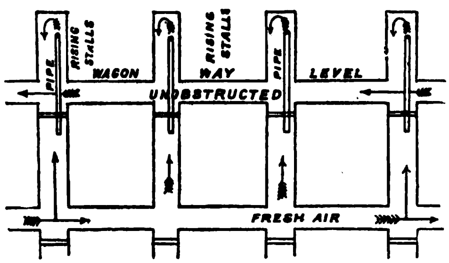


FIG. 278.—Air-pipes for a number of stalls.

level to the face of heading; there will thus be a separate supply of fresh air blowing into each heading.

Doors, Stoppings, etc.—It is frequently necessary to make, as in the above example (Fig. 277), a passage through brattice, and the opening is closed by a door. This door is usually in the form of a piece of brattice cloth nailed at the top and loose at the bottom, so that it can be partly lifted and partly pushed on one side.

In driving a pair of roads, as each new cross-cut is made the old one requires to be stopped up; a permanent stopping may be made by filling or half filling the cross-cut with dirt, or by building a brick wall with mortar 9 inches thick. A stopping composed of a single wall would be broken down in case there was an explosion, but a stopping composed of a heap of dirt 4 or 5 yards long at the top would not be cleared out. Stoppings are frequently made with a brick wall at one end of the place, which is afterwards partly or entirely filled with dirt. In fiery seams it is desirable that the stoppings should not be quite tight, so that some air may always draw through; otherwise each partially filled cross-cut would become a little reservoir of explosive gases. A temporary stopping may be made by nailing boards across, or, more usually, by nailing cloth across. If it is necessary for the conveyance of mineral or the passage of workmen to have a door, this may be made by hanging a sheet; such a door is necessarily leaky, and it is usual to hang another sheet at a distance of a few yards. Sometimes three such sheets may be hung in one cross-cut; each of these canvas doors may be made of a double sheet; the opening is in the middle. Where there is a considerable pressure of air, it is necessary to have wooden doors to prevent a large leakage. These doors are generally fixed in a wooden frame, and built into a brick stopping. As a general rule, every door should be checked, so that when it is open there will be no through current; that is to say, there must be a second door which will be shut all the time the first is open, and this second door must be at such a distance that a horse with a train of waggons may rest between the two doors when closed. Sometimes a sheet is hung instead of a check door. Where the doors separate the main intake from the main return, there should be at least three doors, each separated by a sufficient distance for a horse and train of waggons. In roadways connecting the intake and return near the pit-bottom, the separation may be maintained by four or five doors. In case of an explosion, it is probable that all these doors would be shattered and swept away. To meet this contingency, reserve doors are sometimes attached to the roof and fastened open with a latch. It is thought that the blast would pass under such a door, so that after the explosion it might be let down. It is often necessary to divert the main portion of an air-current by doors, whilst allowing some to pass; a sliding shutter is then placed either in the door or in the stopping (see Fig. 279); this slide can be adjusted so as to leave an opening sufficient and then fastened. These sliding shutters are technically called regulators. A regulator is very conveniently made by hanging sheets across the road so as to hinder the current; these sheets are of comparatively little hindrance to the passage of horses, etc., and are less liable to be left open. Every door must be so hinged that after it has been opened it will swing to automatically. Owing to the squeeze

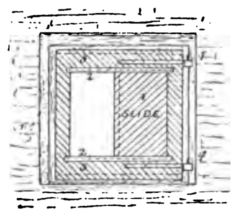


FIG. 279.—Regulator or scale door. 1, sliding door; 2, grooves in which door slides; 3, door swinging on hinges 4.

of the strata, the door-frames are constantly pushed out of shape and position, necessitating continual readjustment of the door. When a driver approaches a door, he has to leave his horse to prop the door open, and it is evident that it may occasionally, if rarely, happen that the door is so left open. Where there is much traffic it is necessary to have a door-boy to open and close the door; this is the proper arrangement, but for economy of working it is necessary to have very few doors on the haulage roads. It is very frequently necessary to take one air-road across another; this is done by air-crossings, sometimes called overcasts. The road which serves for haulage is continued at the right level or gradient; the road which is merely for ventilation is taken over or under. Fig. 280 shows an overcast of simple contrivance. Fig. 281 shows one composed of brick and timber, the construction of which will be understood from the drawing. Two small doors are shown leading into a covered passage, and

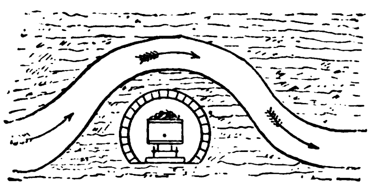


FIG. 280.—Overcast in ground.

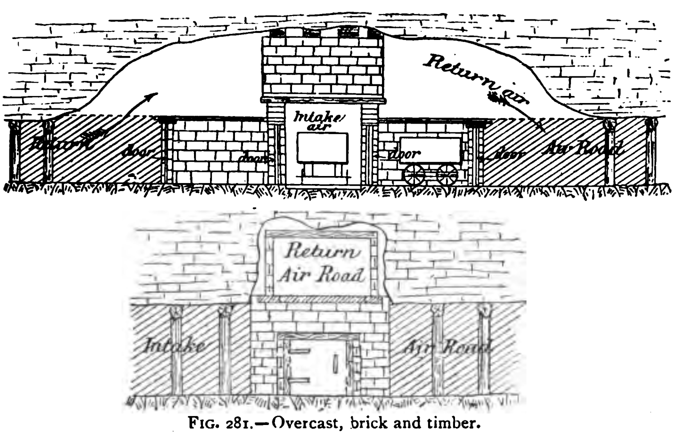


FIG. 281.—Overcast, brick and timber.

thence into the return air-way; through these waggons can be taken to use in air-road repairs. Fig. 282 shows an air-crossing made with iron; and Fig. 283 shows a temporary air-crossing made by building two stoppings in the return and connecting them by pipes laid across the horse-road. There must, of course, be a door in the stopping, or near, to allow men to pass, if necessary. The use of overcasts is essential to the proper ventilation of a large mine. Fig. 284 shows the arrangement of ventilation at a colliery, which is sufficiently explained by means of the references. It will be seen that there are four main air-currents, or splits as they are

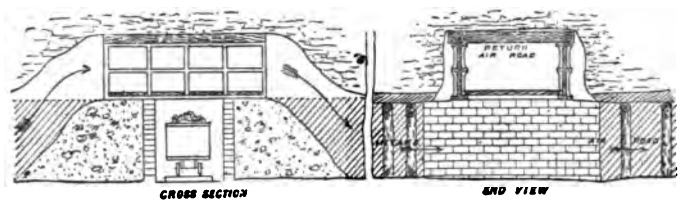


FIG. 282.—Overcast, iron.

technically called; each of these splits is divided again, making nine in all, so that fresh air is brought to the workings in nine different places. Some of these air-currents traverse a shorter distance than others, and an undue proportion of ventilation would go into that district were it not for the regulator shown on the plan. It is sometimes practicable, with a little care, to arrange that the ventilating currents shall be nearly the same length; in that case regulators are not required, and the air will divide itself evenly throughout the mine if all the airways are the same size. A short difference in length does not materially affect the amount of ventilation, because the amount of ventilation in all air-courses of different lengths varies, not in proportion to their lengths, but in proportion to the square root of their lengths. Thus if one road were 10,000 feet long, and the other road were 6400 feet long, each road being of the same sectional area and connected with the same shafts, then the air-current in each would be inversely proportional to the square roots of 10,000 and 6400; that is to say, in the proportion of 100 to 80, the larger quantity passing, of course, along the shorter road. If it were desired to make the ventilation more nearly equal in each district, a few sheets hung partly across the shorter road would have that effect. It will be seen, on reference to the plan, that each main split of

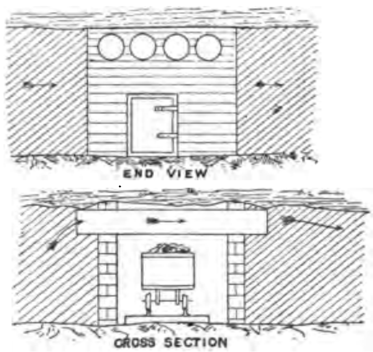


FIG. 283.—Overcast pipes.

the first road requires an overcast, so that three overcasts are required.

The ventilation of metalliferous mines should be conducted on the same principles as those of coal-mines, but it has frequently been very much neglected. The absence of fire-damp, which has but rarely been discovered in metalliferous mines, has

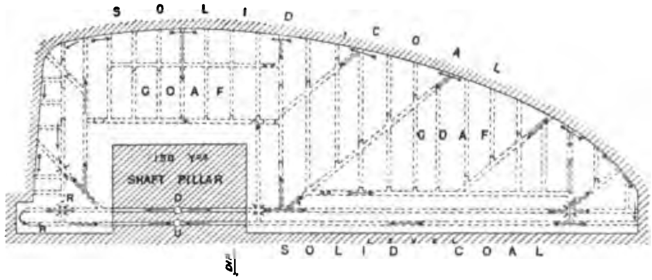


FIG. 284.—Method of working and ventilating a long-wall colliery. ═, roadways in solid coal; ====, roadways in goaf; ⌟⌞ ⌝⌜, overcasts; D, doors; I, stoppings; C, cloths; R, regulator.

removed the main incentive to good ventilation, and miners have been too frequently allowed to work in an atmosphere unfit to breathe. Where the seam is very steep, as is usual in metal-mines, the cross-cuts are generally pits called "winzes." By means of sheets, doors, and stoppings, the air-current can easily be directed. A good air-current is necessary for the proper ventilation of a metal-mine, in order to clear away the smoke from explosives, as well as to supply the men with fresh air to breathe.

Means of producing Ventilation.—No work is ever done without absorbing power, and some source of this must be discovered in every instance. In the case of mines, this power is sometimes in the wind. At the top of the pit is placed a large funnel (see Fig. 285), which is directed towards the quarter from which the wind blows, and the pressure caused by the converging currents in the funnel forces the air down the pipe into the mine. This method is employed in little mines sunk for purposes of exploration, such as lead-mines. In the case of a mine yielding fire-damp, this ventilation would be too irregular to be safe. Another source of power is that of falling water. If a pit were

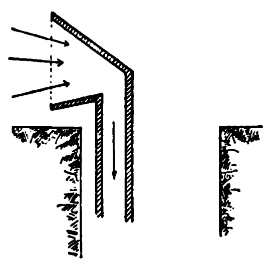


FIG. 285.—Wind-blower.

on a hillside (Fig. 286), and drained by a level from the valley, a stream of water might be turned down the shaft, and in falling, the water would drive a current of air into the mine, the water running off by the level. Another source of power is found in the temperature of the earth. The normal temperature of the earth in temperate latitudes, at a depth of say 50 feet from the surface, is about 50° Fahr., and this temperature is constant winter and summer. Nearer the surface the temperature may vary slightly with the seasons; below this depth it increases. This increase has been found by observation to be in many places at the rate of 1° Fahr. for every 60 feet in depth below the 50 feet normal. Thus at a depth of 650 feet, the temperature of the strata would be 60°; at a depth of 1250 feet, 70°; 1850 feet, 80°, and so on. But this increase of temperature is less rapid, being in some instances 1° for every 90 feet; and in the case of a deep bore-hole in the United States, the temperature was found to increase, down to a depth of 4500 feet, as reported by the underground temperature committee of the British Association, 1892, at the rate of 1° for every 65 feet on the average.

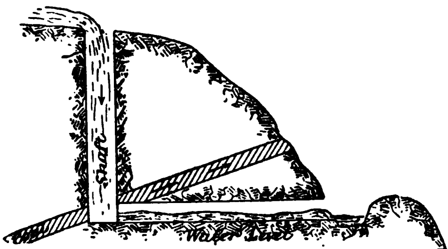


FIG. 286.—Waterfall.

This increase of temperature is proportional to the depth below the surface, and not below the sea-level; thus, if one shaft is sunk at the sea-level to a depth of 1000 feet, and at the same place a level is driven under a mountain 1000 feet high, the temperature in each mine will be the same (see Fig. 287). In the British Isles the temperature of the air at the surface seldom exceeds 60°, and therefore the temperature of a mine 1200 feet deep will, as a general rule, be warmer than the air even in the hottest part of summer.

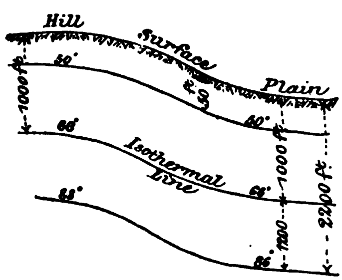


FIG. 287.—Earth-temperature.

In the case of a mine with two shafts, it is nearly always the case that there is a current up one shaft and down the other. Though it may be difficult to say what starts the current, yet, when once started, it is easy to understand why it should continue, because the air in the downcast will have a temperature of say 50°, and will pass through a mine with a temperature of say 70°, therefore the air in the upcast will have a temperature of nearly 70°, and will therefore be lighter than the air in the downcast. This being so, of course it will continue to ascend, and the air in the downcast will continue to descend. It is as if there were a pair of scales, and the cold air were in one scale and the warm air in the other, in equal volumes; the cold air, being heavier, would of course overbalance the light air, and the scale containing the light air would go up. If, instead of a mine 1200 feet deep, a mine only 100 feet deep is considered, then in winter-time, if a current were by some accident started in one direction, it would continue in that direction, because the air in the downcast would be of say 40° temperature, the mine would have a temperature of say 50°, and the air which passes through the mine and entered the upcast would have also a temperature of 50°, and would warm the upcast shaft. This current would continue until the approach of summer, when the air coming to the downcast would also have a temperature of 50°, and the mine is no longer increasing the temperature of the air; therefore, the downcast being as warm as the upcast, there is no difference of weight, and therefore no current, and the mine would be unventilated, and work would have to be suspended. In summer-time, however, there may be natural ventilation in case the two shafts are not the same depth. Suppose one shaft was sunk on a hill to a depth of 150 feet, and the other shaft was sunk in the valley 50 feet deep, the difference in level between the two tops of the two pits being 100 feet (see Fig. 288); then a current would start down the deep pit, because the ground would cool the air, and it would be heavier than the free air above the top of the upcast, and the current of air would not cease until the free air was as cold as the air in the downcast shaft.

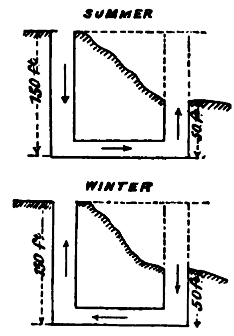


FIG. 288. — Air-currents in shallow pits, summer and winter.

The uncertainty and changeableness of natural ventilation in shallow mines has led to the use of furnaces, by which the temperature of the air entering the upcast is increased, so that even in summer-time it is hotter than the downcast air; but in deep mines, where the natural ventilation is constant all the year round, it is still insufficient to cause a powerful current through a mine of great extent.

In metal-mines the workings do not so often extend to great distances from the shaft as in coal-mines, and therefore natural ventilation is more common, but in collieries it is now exceedingly rare.

A simple fire-lamp, as is shown in Fig. 289, is sometimes used; the lamp is filled with coal, and then lowered down the pit, and has to be raised up again when more coal is required. This is a convenient arrangement sometimes. A more permanent and satisfactory arrangement is shown in Fig. 290. If the shaft is used for winding men or materials, the furnace should not be too close, or the flames may reach into the shaft; and the smoke-drift from the furnace should enter the shaft not less than 20 or 30 yards from the bottom, in order that, when men are being lowered down, they may pass the smoke-drift at considerable speed. Whereas, if the furnace-drift were close to the pit-bottom, the heat would be too great to bear whilst the cage was being slowly lowered to the ground.

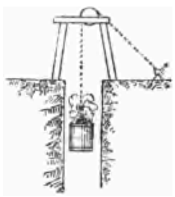


FIG. 289.—Fire-lamp in shaft.

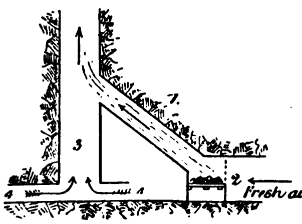


FIG. 290.—Furnace and dumb-drift. 1, furnace-drift; 2, furnace fed with fresh air; 3, shaft; 4, main returns.

Fig. 291 shows a design for single furnace, the internal arch 6 feet wide, and the bars 6 feet long; the internal arch is made of fire-brick; the external arch is red brick. There is a clear space between the arches from 9 inches to 1 foot in width; the current of air continually passes through these spaces. By means of this current the heat of the furnace is prevented from passing to the outer ring of brickwork, and through

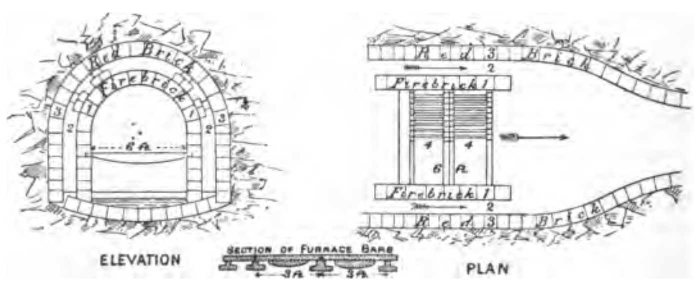


FIG. 291.—Ventilating furnace. 1, inner lining of firebrick; 2, space for current of air· 3, outer lining of common brick; 4, furnace bars and supports.

that ring to the strata, to which it might set fire. The heat of the smoke-drift is not dangerously high, because of the amount of air mixed with the smoke.

Fig. 292 shows a double furnace with two fire-brick arches and a large red-brick arch over all. The air-current passes along

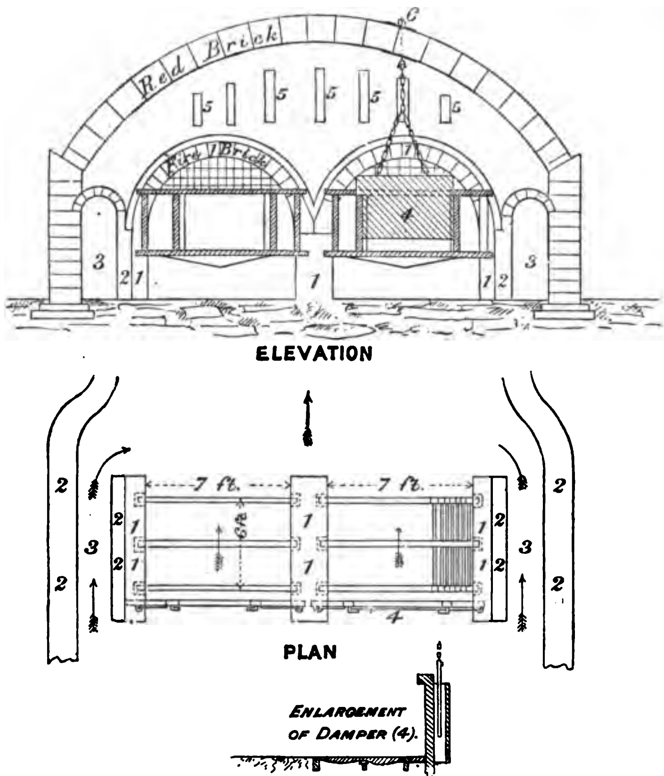


FIG. 292.—Double ventilating furnace. 1, fire-brick linings; 2, common red brick; 3, man-hole and air-passages; 4, damper doors; 5, air-holes; 6, arrangement for adjusting damper door.

the sides and over the top of the fire-brick arches; the lifting doors shield the furnace-man from the heat of the fire. Whilst one of the furnaces is being cleaned, the other is at its greatest heat, so that the ventilation is never slackened.

Fig. 293 shows a ventilating furnace with a length of bars of 30 feet. It is fired at five side doors; by means of these numerous fire-holes and great length of bars, a regular and intense heat can be maintained. In some cases where great ventilating power is required, the furnace is fired from both sides. In many mines it is not considered safe that the air from the workings or from some portions of the workings should pass through the furnace, because it might be mixed with explosive gas. In this case the furnace is fed, either with air from the downcast, or from some district which is considered to be quite safe from blowers of gas; the rest of the air is conveyed by a separate road with a dumb-drift into the shaft either below the smoke-drift or above it, usually below. In this case the air from the workings is heated by mixture with the hot air and smoke that comes from the furnace. When the mine air enters the shaft below the furnace-drift, it can

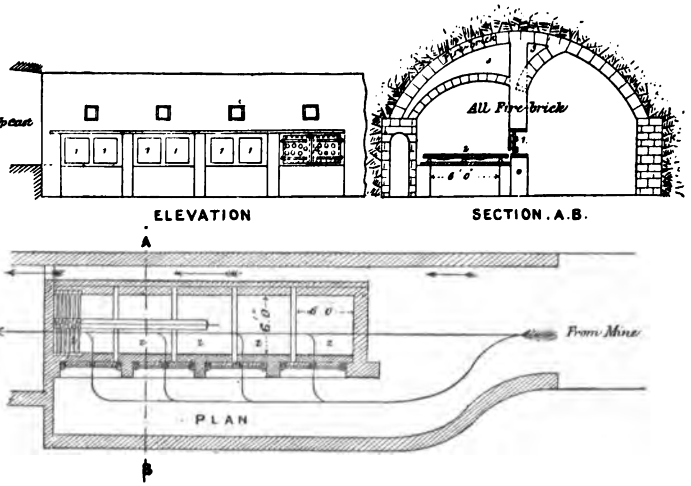


FIG. 293.—Range of four furnaces. 1, fire-doors; 2, fire bars and grates; 3, air-spaces.

be used as a winding shaft. In case, however, the dumb-drift should enter the upcast above the furnace, then the heat of the shaft below the dumb-drift and above the furnace-drift would be too great, perhaps 500°, so that the shaft could not be used as a winding shaft, nor would it be safe for any person to enter the shaft unless the furnace were quite out.

Thirty years ago, ventilation of collieries by furnaces was the rule, and there were only two or three mines in the country ventilated by fans. At the present time, furnaces are in the minority at first-class collieries, though they are still numerous.

The objections to a furnace are, first, the possibility of danger from a blower of gas; secondly, the inconvenience of shaft repairs, if the upcast is used for winding; thirdly, the expense of the fuel and labour of furnace-men; fourthly, the corrosion of iron tubbing and damage to it caused by the expansion and contraction due to a varying temperature. In some cases, however, it is convenient to have steam boilers in the pit, and the waste heat from these boiler fires, if turned into the upcast, is a most economical way of producing ventilation; and where there is no danger from fire-damp, this is a very good system. The deeper the mine the more efficient is the furnace. In a dry shaft the heat escapes but slowly through the brickwork, and therefore the heat, which is put into the air at the bottom of the upcast, will, for the most part, remain in it till it gets to the top.

The ventilating power of an upcast shaft depends on the difference in weight between the air in it and the air in the downcast shaft. Thus, in the case of a mine 1000 feet deep, for given temperatures there will be a given difference in weight; but if the depths of both shafts are increased to 2000 feet, the average temperature of each remaining the same, then the difference in the weights will be doubled, and consequently the ventilating power of the upcast will be doubled; but the amount of fuel required to heat the air in the upcast will remain the same if the quantity of air passing is the same.

Table VIII., p. 204,[1] shows the weight of air at various temperatures and barometrical pressures, and also the expansion of air at equal pressures with increased temperature.

By means of this table, the difference in weight between two columns of different temperature can be quickly ascertained. It will be useful to bear in mind that, for the purposes of furnace and of mechanical ventilation, the depth of the downcast shaft is immaterial; the depth of the upcast only is important.

[1] J. W. Thomas, F.C.S., "Treatise on Coal-Mines, Gases, and Ventilation."

Table VIII.

Temperature on Fahrenheit's scale.	Weight of 100 cubic feet of air in pounds, barometer 29 inches.	Difference between 10° in temperature in parts of a pound.	Weight of 100 cubic feet of air in pounds, barometer 30 inches.	Difference between 10° in temperature in parts of a pound.	Weight of 100 cubic feet of air in pounds, barometer 31 inches.	Difference between 10° in temperature in parts of a pound.	Difference in weight of 100 cubic feet of air due to a rise or fall in the barometer of 1 inch.	Expansion of 100 cubic feet of air by being elevated in temperature 10° Fahr.
30	7·838	—	8·108	—	8·378	—	0·270	—
32	7·805	—	8·074	—	8·343	—	0·269	100·00
42	7·680	0·158	7·945	0·163	8·210	0·168	0·265	102·04
52	7·530	0·150	7·790	0·155	8·050	0·160	0·259	104·07
62	7·387	0·143	7·641	0·149	7·895	0·155	0·254	106·11
72	7·247	0·137	7·497	0·144	7·747	0·148	0·250	108·14
82	7·113	0·134	7·358	0·139	7·603	0·144	0·245	110·18
92	6·984	0·129	7·225	0·133	7·466	0·137	0·241	112·22
102	6·861	0·123	7·097	0·128	7·333	0·133	0·236	114·25
112	6·740	0·121	6·972	0·125	7·204	0·129	0·232	116·29
122	6·624	0·116	6·852	0·120	7·080	0·124	0·228	118·32
132	6·512	0·112	6·736	0·116	6·960	0·120	0·224	120·36
142	6·403	0·109	6·624	0·112	6·845	0·115	0·221	122·39
152	6·299	0·104	6·516	0·108	6·733	0·112	0·217	124·43
162	6·198	0·101	6·411	0·105	6·624	0·109	0·213	126·47
172	6·099	0·099	6·309	0·102	6·519	0·105	0·210	128·50
182	6·003	0·096	6·210	0·099	6·417	0·102	0·207	130·54
192	5·910	0·093	6·114	0·096	6·318	0·099	0·204	132·58
202	5·820	0·090	6·023	0·093	6·223	0·095	0·201	134·62
212	5·735	0·087	5·933	0·090	6·131	0·093	0·198	136·66
222	5·651	0·084	5·846	0·087	6·041	0·090	0·195	138·69
232	5·569	0·082	5·761	0·085	5·953	0·088	0·192	140·73
242	5·490	0·079	5·679	0·082	5·868	0·085	0·189	142·76
252	5·413	0·077	5·600	0·079	5·786	0·082	0·186	144·80
262	5·338	0·075	5·522	0·078	5·706	0·080	0·184	146·84
272	5·265	0·073	5·446	0·076	5·627	0·079	0·181	148·88
282	5·193	0·072	5·372	0·074	5·551	0·076	0·179	150·92
292	5·124	0·069	5·300	0·072	5·477	0·074	0·176	152·96
302	5·057	0·067	5·231	0·069	5·405	0·071	0·174	155·00

Fig. 294 shows a mine where the fresh air enters by a level drift, and leaves the mine by a vertical shaft 1000 feet in depth. The dotted lines above the entrance to the intake show a hypothetical downcast the same depth as the upcast. In every mine where there is a powerful ventilation, the temperature of the air in the downcast differs very little from that of the air on the surface, and therefore the pressure of a column of air at the bottom of the downcast is the same per square foot, whether it is confined within the walls of a shaft or is unconfined as in the figure. (In the case of a downcast shaft there would be some reduction of pressure due to friction against the shaft-sides.)

Fig. 295 shows an upcast shaft 500 feet deep, and a downcast shaft 1000 feet deep, the bottom of each shaft being on the same level. In this case there is a height of 500 feet of fresh air above the top of the upcast, which is equal in weight to the first 500

FIGS. 294, 295.—Hypothetical downcasts.

feet of the downcast shaft, and therefore these two upper parts of the column may be disregarded, and the effect of the downcast is as if it were 500 feet deep, that is, the same depth as the upcast. For future calculations, therefore, the upcast depth alone will be considered, the downcast depth being no more important than an equal length of level road.

Ventilating Pressure produced by Furnaces.—The following instance shows a method of calculating ventilating pressure. The upcast shaft is 1000 feet high above the smoke-drift, and has an average temperature of 192° Fahr.; the down-cast or intake temperature is 62° Fahr.; the barometer half-way down the shaft is 30 inches. On referring to the table, it will be seen that the weight of a column of air 1000 feet high and 1 square foot in sectional area at 62° is 76·41 lbs., and a similar column at 192° is 61·14 lbs., the difference being 15·27 lbs. This, therefore, is the unbalanced weight for each column 1 foot square at the base.

Fig. 296 shows the effect of a furnace in a shaft 1000 feet deep.

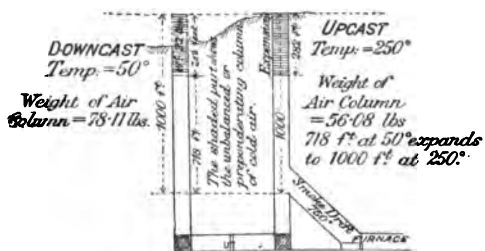


FIG. 296.—Pressure and temperature in furnace-pit, dumb-drift.

Water-gauge.—The ventilating pressure in mines is not generally measured in pounds, but by a water-gauge. This water-gauge is usually made of a glass tube from ½ inch to 1 inch bore, bent in the form of the letter U (see Figs. 296 and 296*a*). A scale divided into inches and tenths is placed between the two columns. One tube is covered with a metal cap perforated to admit the atmospheric pressure; the other has also a metal cap with a branch, 2 or 3 inches long, which can be put through a hole in a door or other partition: thus one column is connected with the air on one side of the partition, and the other column with the air on the opposite side of the partition. If there should be any difference of atmospheric pressure on these two columns, the water which is in the tube will measure this difference; if the water on one side is 1 inch higher than the water on the other side, there is a pressure of 1 inch of water-gauge. One inch of water-gauge is equal to a pressure of 5·2 lbs. on each square foot; this is evident, because a cubic foot of water weighs about 62·4 lbs., and $\frac{1}{12}$ of a cubic foot therefore weighs about 5·2 lbs.

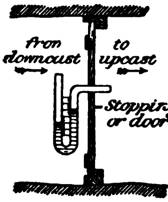


FIG. 296*a*.—Water-gauge.

Ventilating Pressure produced by Furnaces (*continued*).—Referring to the instance above given, where the ventilating pressure is 15·27 lbs., if this is divided by 5·2 the inches of water-gauge will be found. In this case the water-gauge pressure is 2·93 inches, or, in round figures, 3 inches.

The pressure may also be calculated in another way without reference to the table of weights above given. For the purposes of approximate calculations, it is sufficient to remember that water is about 800 times as dense as air, or that 1 cubic foot of water is about as heavy as 800 cubic feet of air at a temperature of 52° and barometer 30 inches; the exact proportion, of course, varies with the temperature and pressure.

In the case of air and water at a temperature of 62° Fahr. and a pressure of 30 inches, 1 cubic foot of water is about as heavy as 820 cubic feet of air. When air is heated and is not confined, it expands. 459 cubic feet of air, at a temperature of zero Fahr., will expand 1 cubic foot if raised 1° in temperature, and will therefore occupy 460 cubic feet, and an equal expansion will take place for every degree of temperature. Thus, if the temperature were raised 41°, the expansion would be 41 cubic feet, or the 459 would be increased to 500; and if the temperature were raised 141° the 459 would be increased 141 cubic feet, or the 459 cubic feet would be expanded to 600 cubic feet. Bearing in mind the above statements, the following rule will be understood:—

D = depth of upcast in feet.
T = temperature of upcast in degrees Fahrenheit.
t = temperature of downcast.
T′ = temperature of upcast + 459.
t' = temperature of downcast + 459.
x = ventilating column in feet of air of the downcast temperature.

$$\text{Then } x = D - \frac{D \times t'}{T'}$$

$$\text{and WG} = \text{water-gauge pressure in inches} = \frac{x \times 12 \times 1{\cdot}22}{1000}$$

$$P = \text{pressure in pounds per square foot} = \text{WG} \times 5{\cdot}2$$

$$P = x\frac{76}{1000} \text{ (approximately)}$$

$$Q = \text{quantity of air in cubic feet per minute.}$$

$$\text{FP} = \text{foot-pounds of work done per minute} = Q \times P$$

$$\text{HP} = \text{horse-power} = \frac{Q \times P}{33{,}000}$$

Let T = 193° Fahr., and t = 60°
D = 1000   Q = 100,000.
T′ = 193 + 459 = 652.
t' = 60 + 459 = 519.

Then, applying the rule—

$$x = (D)1000 - \frac{(D)1000 \times (t')519}{(T')652} = 204$$

Thus the ventilating column is 204 feet of air at a temperature of 60° and barometer 30.

$$\text{WG} = \frac{(x)204 \times 12 \times 1{\cdot}22}{1000} = 2{\cdot}98 \text{ inches}$$

Thus the ventilating pressure is equal to, say, 3 inches of water-gauge.

$$P = \text{WG} \times 5{\cdot}2 = 15{\cdot}6 \text{ lbs.}$$

$$\text{FP} = (Q)100{,}000 \times (P)15{\cdot}6 = 1{,}560{,}000 \text{ foot-pounds}$$

$$\text{and HP} = \frac{(Q)100{,}000 \times (P)15{\cdot}6}{33{,}000} = 47{\cdot}27$$

or the horse-power of the upcast shaft is about 47¼.

The student is, of course, aware that a horse-power is equal to 33,000 foot-pounds a minute—that is to say, is equal to work done in lifting 33,000 pounds 1 foot high a minute, or equal to lifting 1 lb. 33,000 feet in a minute.

To ascertain the actual pressure of the ventilating column, an addition must be made to that recorded by the water-gauge at the bottom of upcast to allow for the resistances in the shafts; the total pressure can be best ascertained by calculation founded on careful observations of the shaft temperatures and barometic pressure.

Fuel required for Furnace Ventilation.—The amount of fuel necessary to heat the upcast shaft will depend partly upon the completeness of combustion. If a certain amount of the fuel is raked out of the fire with the ashes, or passes up the shaft as unburnt smoke, there will be a corresponding increase in the amount required; if the shaft is wet, or if the upcast is tubbed, and there is a circulation of water at the back of the tubbing, by which heat can be carried away from the shaft, then more fuel will be required than in a perfectly dry brick-lined shaft.

For the purposes of theoretical calculation, it is assumed that the shaft is quite dry, and that no heat is lost by conduction through the strata; it is also assumed that the air will reach the ventilating furnace at the downcast temperature. Both these assumptions are different from the actual facts. In the first place, the shaft is always somewhat damp, unless artificially dried; in the second place, there is always some conduction of heat through the strata; in the third place, in mines of 1000 feet and upwards in depth, the air is always warmer after passing through the mine than the downcast air. But this heat gained from the mine may be set against the heat extracted from the air by the sides of the upcast, and it is not improbable that one will balance the other where there is no water in the upcast except what is properly guided in pipes.

The following suggestions will show that this is not an unreasonable assumption. The rate at which heat passes through a conductor is proportional to the difference in the temperatures on each side.[1] Thus in the upcast the temperature of the strata would be say 60°, and of the air say 180°; in the air-ways of the mine the average temperature would be say 60°, that of the rocks in contact say 65°. Therefore the transmission of heat in the upcast, as compared with the transmission from the sides of the mine, to the air-current would be as 120 : 5, or 24 : 1. Therefore, if the upcast shaft is 1000 feet deep, as much heat will be lost in that length as is gained in 24,000 feet of underground air-roads.

The heat produced by the combustion of 1 lb. of coal is called its calorific power, and is equal to 14,400 units; a unit is 1 lb. of water raised 1° Fahr. This calorific power is for pure coal and perfect combustion. For the purpose of a colliery furnace the calorific power may be assumed at 12,000 units only.

It takes less heat to raise 1 lb. of air 1° than to raise 1 lb. of water 1°.

[1] The heat passing through any substance varies directly as the difference of temperature of the two surfaces (Box on "Heat," p. 210).

The relative heat required to heat a body 1° is called its specific heat; thus the specific heat of water is 1, but the specific heat of air is 0·2374.

Therefore the weight of air that 1 lb. of coal will raise 1° Fahr. is found by the following rule-of-three sum:—

0·2374 : 1 : : 12,000 : 50,547, or say 50,000 lbs.

To take a case. The ventilation is 100,000 cubic feet a minute, the intake temperature is 50°, the upcast temperature is 130°; the heat required from the furnace is 130 − 50 = 80°; 100,000 cubic feet at 50° weigh 7811 lbs.; therefore the fuel required per minute $= \frac{7811 \times 80}{50{,}000} = 12{\cdot}49$ lbs.; multiplying this by 60, the answer, 749·4, represents the pounds of fuel required per hour.

Ventilating pressure is 10·61 lbs.; F.P. = 1,061,000 foot-pounds; and H.P. = 32·15.

749·4 lbs. per hour divided by 32·15 = 23·3, the fuel consumed per horse-power per hour,[1] or say 24 lbs. of fuel are required per horse-power in a mine 1000 feet deep. As before stated, the efficiency of the furnace varies directly as the depth, and the fuel consumption, therefore, per horse-power varies inversely as the depth, and may be shown in the following table:—

Shaft	250 feet deep:	full consumption	per	H.P.	96 lbs.
,,	500	,,	,,	,,	48 ,,
,,	1000	,,	,,	,,	24 ,,
,,	1500	,,	,,	,,	16 ,,
,,	2000	,,	,,	,,	12 ,,
,,	3000	,,	,,	,,	8 ,,
,,	4000	,,	,,	,,	6 ,,

So far as published statements (see Table IX.) are concerned, the actual fuel consumption is sometimes more and sometimes less than the figures given in the above theoretical statement.

[1] Fuel consumed is generally taken at the amount per hour per horse-power.

TABLE IX.

	Authority.	Depth of upcast in feet.	Temperature.	Coals burnt per diem	Ventilation in cubic feet.	Ventilating pressure in pounds per square foot.	Pounds of coal consumed per H.P. per hour as reported.	Calculated[1] theoretical minimum quantity of fuel required per H.P., pounds per hour.
				tns. cwt.				
Furnace	R. Howe	260	240	3 12	30,358	5·376	68	92·3
,,	,,	655	117	3 1	48,230	5·263	37	36·6
,,	,,	234	198	—	50,071	4·036	66	102·5
,,	C. Houfton	345	200	12 0	57,772	5·660	113	69·5
,,	,,	600	85	3 0	52,000	2·880	61	40·0
,,	W. Cochrane	1380	—	—	126,366	11·180	26	17·3
,,	,,	798	—	—	99,750	9·620	29	30·07
,,	J. Williamson	594	130	5 0	55,120	6·77	18½	40·4
,,	Particulars reported to the author by three colliery managers.	900	(some steam also)	8 0	280,000	9·10	9½	26·6
,,		1200		6 10	100,000	8·64	23	20·0
,,		870		8 0	156,000	9·88	16	27·5

[1] In this calculation no allowance is made for the heating of the air by the strata of the mine, nor for cooling by water or conduction in the shaft.

If all the air from the mine enters the shaft by a dumb-drift, and the furnace is entirely supplied with fresh air, the economy of furnace-ventilation is much reduced, because the fresh air passing through the furnace serves no useful purpose except to transmit the heat of the fire to the air in the upcast. In the case of a dumb-drift, if the temperature of the smoke-drift is 500°, and that of the mine air 50°, and the average temperature of the mixture above the furnace-drift 200°, then each pound of air from the furnace will have 300° of temperature to spare for heating the mine air. And each pound of mine air will lack 150° of the upcast temperature, therefore 1 lb. of furnace air will heat 2 lbs. of mine air; and of the total weight of air in the upcast, 2 out of 3 lbs. will be mine air, and 1 out of 3 lbs. will be furnace air; that is to say, the weight of mine air is increased 50 per cent. by the addition of the furnace air, and consequently the amount of fuel required will be also increased 50 per cent. of the quantity required if the mine air passed through the furnace.

Another method of calculating the relative quantities of mine air, fresh air, and furnace air passing up the shaft is as follows:

Furnace-ventilation with a dumb-drift. Upcast, 11 feet diameter; downcast, 9 feet diameter.

Rule.	Example.
F = temperature of furnace-drift	= 500°
T = " " upcast	= 200°
t = " " mine air	= 65°
V = volume of mine air	= 100,000 cub. ft.
v = " fresh air and smoke at 500°	= 82,356 "
v' = " " " reduced to 200°	= 56,593 "
V' = " mine air expanded to T°	= 125,763 "
V'' = total volume in upcast = $V' + v'$	= 182,356 "

$$V' = V \times \frac{459 + T}{459 + t} = 100{,}000 \times \frac{459 + 200}{459 + 65}$$

$$= 100{,}000 \times \frac{659}{524} = 125{,}763 \text{ "}$$

$$v' = V' \div \frac{F - T}{T - t} = 125{,}763 \div \frac{500 - 200}{200 - 65}$$

$$= 125{,}763 \div \frac{300}{135} = 56{,}593 \text{ "}$$

$$v = v' \times \frac{459 + F}{459 + T} = 56{,}593 \times \frac{459 + 500}{459 + 200}$$

$$= 56{,}593 \times \frac{959}{659} = 82{,}356 \text{ "}$$

$$V'' = V' + v' = 125{,}763 + 56{,}593 = 182{,}356 \text{ "}$$

Mechanical Ventilation.—This means the use of pumps or fans to produce an air-current, instead of a furnace. The mechanical ventilator of a colliery is, with rare exceptions, at the top of the upcast, which is covered over; a drift or tunnel from the shaft conveys the air into the fan-chamber. The covering at the top of the upcast may be movable, so as to permit the passage of cages. This covering is often made by a simple bonnet of wood or iron as light as possible, which is lifted up by the cage as it rises, and dropped on the platform as the cage descends. The entrance to the shaft at the top is divided into two compartments, just sufficient for the cage to pass through, and as deep as the cage, or at least as deep as from the top of the cage to the first platform (see Fig. 297). Thus, when the bonnet is lifted, the platform of the cage blocks the entrance to the shaft, and prevents any large inrush of air. Sometimes the whole pit-frame is covered in nearly as high as the pulley wheels, and a partition separates the two cages. When the cage has been raised to the pit-top, doors in the covering are opened, through which the empty and full waggons can be run (see Fig. 298).[1] It is not found that the ventilation of the mine is materially injured by the opening and closing of the shaft-covering.

In considering the economy of mechanical ventilation, the term "efficiency" is generally used, and is given in percentages;

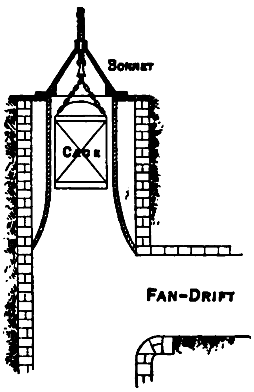


Fig. 297.—Bonnet for fan-shaft.

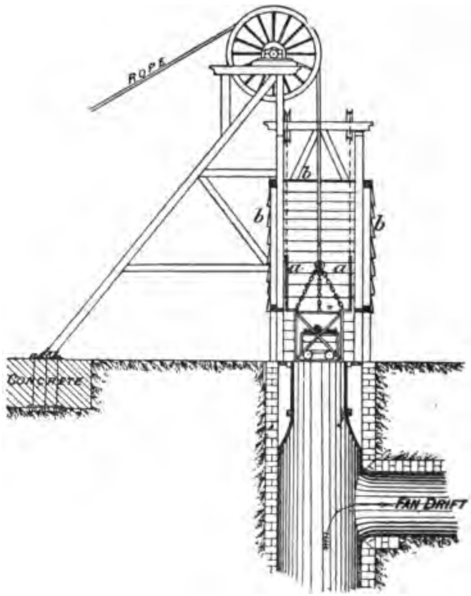


Fig. 298.—Cover for fan-shaft.

thus the horse-power of the engine is taken as 100 per cent., and the air in the shaft is something less, and may be anything between 10 and 80 per cent.

The horse-power of the engine is ascertained by means of the indicator (see Fig. 299). The indicator shown in the figure is that known as the Richards, and is the one most commonly used.

Other indicators, such as the Crosby, or Tabor, are now preferred by many as being better for high speeds, and in other respects. The Richards indicator has a piston working in a small brass cylinder.

[1] In this figure the doors are on slides, and their weight reduced by balance-weights; the cage lifts them when it comes up to the top. *a a* are the doors, and *b b b* is the wooden covering which prevents the air entering the shaft. When the cage is lowered, the sliding doors *a a* drop down on to the top of the pit and close up the opening.

The piston is held down by a steel spring; to the upper end of the piston-rod are fastened levers, one of which carries a pencil. The piston raises the pencil up and down in a directly vertical line. The indicator is screwed on to one end of the cylinder, through which a half-inch hole is drilled (see Fig. 300). The pressure of the steam in the cylinder underneath the indicator-piston compresses the spring and raises the pencil. The pencil is pressed against a piece of paper on a drum. As the piston of the steam-engine moves, the drum is turned by a cord connected with the cross-head, and reducing-gear. By this means the pencil draws a diagram. If there is no steam-pressure in the cylinder, this diagram is a simple, straight, horizontal line, which is called the atmospheric line. When steam enters the cylinder, the pencil is forced up, and shows the amount of pressure by the height to which it goes. As the engine moves, the line traced on the diagram would be horizontal if the steam-pressure were constant; if the steam-pressure falls, the line also falls. Thus the pressure at any part of the stroke is indicated

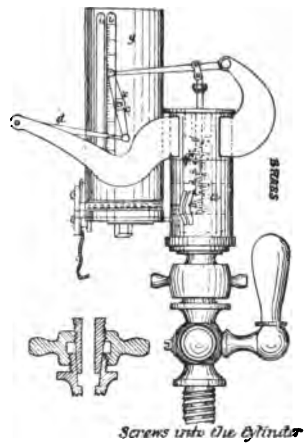


FIG. 299.—Richards's indicator.

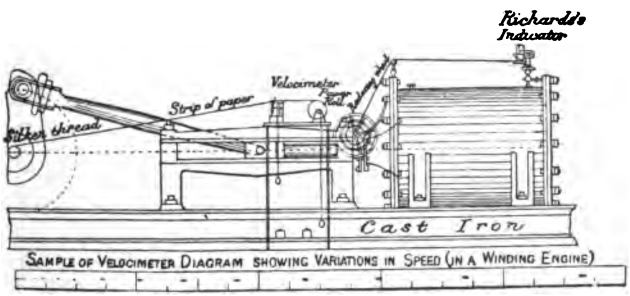


FIG. 300.—Horizontal steam-engine, showing the mode of applying Richards's indicator and the velocimeter.

by the height of the line above the starting-point. When the piston comes back, it ought to fall to the level of atmospheric pressure in a non-condensing engine; if it does not, it is because the steam has not been completely exhausted. In a condensing engine, it falls below the atmospheric line. The height of the return line above the starting-point is the measure of the back pressure in the cylinder.

Fig. 301 shows four indicator diagrams, taken from two engines, one from each end of the cylinder. As is apparent from the figure, one of the engines is condensing, and the other non-condensing. An indicator must be placed at each end of the cylinder, because it is not likely that both ends will give precisely the same diagram. The average pressure may be measured by dividing the diagram with parallel lines into 10 equal spaces. The distance between the steam line and the back-pressure line is measured with a scale made to suit the spring, and the total measurements divided by 10 will give the average pressure. The averages on each end of the cylinder are

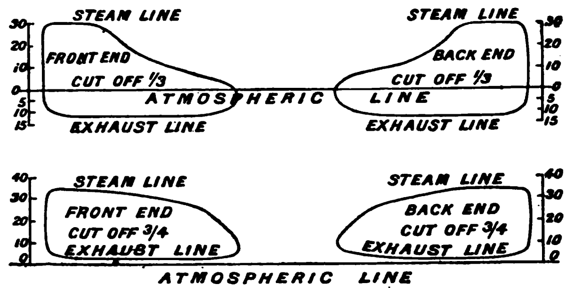


FIG. 301.—Richards's indicator diagrams.

added together and divided by 2. The area of the engine piston in square inches, the length of the stroke in feet, the number of strokes per minute, and the average steam-pressure are multiplied together to obtain the foot-pounds of work per minute. If this result is divided by 33,000, the result is given in horse-powers. In the following example the average steam-pressure is 40 lbs., diameter of cylinder, 20 inches; stroke of piston, 3 feet; revolutions of fly-wheel, 50 per minute. Then the area of the cylinder is 314·16 square inches; from this has to be deducted the section of the piston-rod, which is 7·06 square inches. As the piston-rod is only at one end, this is divided by 2, making a deduction of 3·53, leaving the effective area of piston 310·63, or say 310 square inches.

Then 40 lbs. × 310 square inches × 3 feet × 2 × 50 revolutions
= 3,720,000 foot-pounds per minute
or dividing by 33,000 = 112·7 H.P.

If the engine is a condensing engine, the back-pressure line comes below the atmospheric line, and indicates the pressure above zero. The diagram is measured in the same way; that is to say, the difference between the steam line and the back-pressure line.

Each indicator should be supplied with three or four springs according to the steam-pressure to be measured, each spring having a scale; these scales vary from 8 lbs. to an inch up to 32 lbs. to an inch, and more for very high pressures. In the improved indicators now made a double spring is used, which is found to give better results than the single spring. There is also less weight in the moving parts, which is an advantage, because the weight prevents perfect accuracy in the diagram owing to the momentum. The Richards indicator is usually considered sufficiently accurate for engines up to 100 revolutions per minute and 300 feet piston-speed.

Horse-power in the Air.—This is calculated, in the way already indicated in the case of furnaces, by means of a water-gauge. For mechanical ventilators the water-gauge is placed in the tunnel leading to the fan, so as to ascertain the difference between the external pressure and that in the fan-drift, this difference (when the fan is at the top of the upcast) being entirely due to the action of the fan.

Great care has to be taken to ascertain this difference with accuracy, because the air-current, rushing towards the tube leading to the water-gauge, may blow into the tube, and so cause a less difference to be indicated than really exists. On the other hand, it may blow away from the tube leading to the gauge, and by suction cause a greater difference of pressure to be indicated than really exists. The tube should therefore end in a box perforated with small holes, and so avoid the effects of the air-current.

The foot-pounds of work done on the air are ascertained as follows:—

Q = cubic feet of air per minute.
P = W.G. in inches $\times$ 5·2 lbs.
$Q \times P$ = foot-pounds.
Say $Q = 100{,}000$, and W.G. $= 3$.
Then $P = 15{\cdot}6$
then foot-pounds $= 100{,}000 \times 15{\cdot}6 = 1{,}560{,}000$
and H.P. $= 47{\cdot}27$, say $47\frac{1}{4}$

Anemometer.—The quantity of air passing in a given place is ascertained by measuring the sectional area, and also measuring the speed at which the air passes. Suppose a road 10 feet wide, 5 feet high, sectional area 50 square feet, speed 1000 feet per minute, then the quantity = 50,000 cubic feet per minute. The speed is measured by an anemometer; the usual kind is Biram's (see Fig. 302). This is similar to a windmill. The face of the indicator disc is set precisely opposite to the air-current; the sails are set at an angle of 45°. The pressure of the air on these sails or vanes takes effect, therefore, in two directions—one in the direction of the current, which produces no effect; the other at right angles to this direction, which turns the wheel round. The spindle of the wheel gives movement to toothed gearing in a case; pointers are connected to this gearing, which indicate the number of revolutions made. The first pointer makes one revolution for every 100 feet lineal of air that blows past the wheel; the second pointer makes one revolution for every 1000 feet; the third pointer, one revolution for every 10,000 feet; the fourth pointer, one revolution for every 100,000 feet; and the fifth pointer, one revolution for every 1,000,000 feet; and the sixth pointer one revolution for 10,000,000 feet. Thus if the speed of the air-current is 1000 feet a minute, it will take 10,000 minutes for the sixth pointer to make one revolution—that is to say, 166⅔ hours.

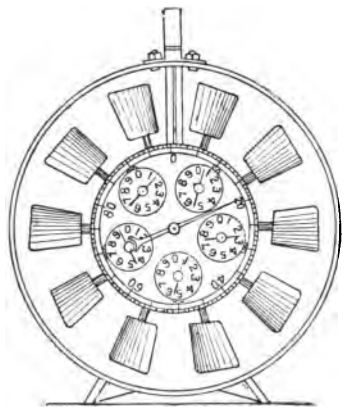


FIG. 302.—Biram's anemometer.

The accuracy of the anemometer may be roughly tested

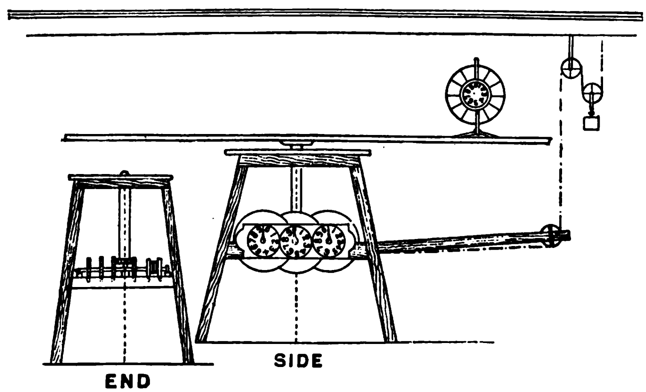


FIG. 302*a*.—Atkinson and Daglish's whirling-machine.

by taking it into a long room or gallery of known length where there is no current of air: holding the face of the anemometer in the direction he moves, and at arm's length, let the operator walk the length of the room. When the finger of the anemometer comes to rest it should indicate the length of the gallery, that being the length of air-current that has passed through the anemometer. Anemometers are generally tested, however, with a whirling-machine (see Fig. 302*a*). The anemometer may thus be moved in a circle of say 30 feet circumference at any desired speed by means of a weight and gearing, and the distance through which it moves is recorded on the dial of the machine. The same distance should be recorded on the anemometer ; if the record is not the same, the difference is noted on a card, to which reference should be made whenever the anemometer is used in order to correct the reading.

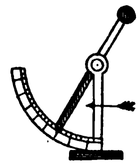


FIG. 303.—Dickinson's anemometer.

Another anemometer (see Fig. 303) is Dickinson's. This is simply a flap which is lifted by the air-current. The height to which it is lifted varies with the speed. The speed is marked on the quadrant.

Another anemometer (see Fig. 304) is Robinson's. In this the arms revolve in a horizontal plane. At the end of each arm is a hemispherical cup. The wind fills the concave surface at full pressure ; it glances off the convex surface of the cup on the opposite arm. The difference in the pressure on the two cups causes the arms to revolve. This kind of anemometer is generally used at meteorological stations ; it is seldom used for mines.

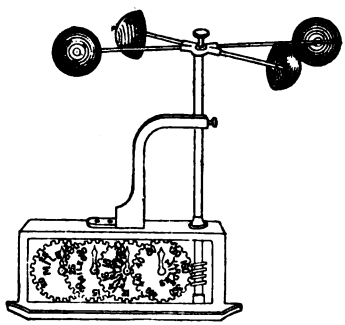


FIG. 304.—Robinson's anemometer.

The speed of the air-current may be measured with powder-smoke. Take an air-road of uniform section, say 100 yards in length. At the intake end let powder be flashed. The observer at the other end, seeing the flash, starts a stop-watch, which he stops the instant he smells the smoke. In this way the time it takes the smoke to travel 100 yards is ascertained.

Another method is to break a bottle containing a strong-smelling spirit, say sulphuric ether. The sound of a hammer or other signal warns the observer that the spirit has been liberated, his nose tells him when the spirit arrives, and his stop-watch shows the time.

Reciprocating Air-pumps.—Fig. 305 shows Nixon's ventilator (Nixon's Navigation, Glamorganshire). This is driven by a horizontal high-pressure steam-engine, one cylinder 36 inches diameter, 6 feet stroke, two fly wheels each fifteen tons. The connecting-rods give movement to two air-pistons, 7-feet stroke, also

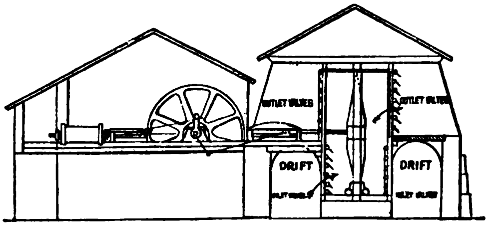


FIG. 305.—The Nixon ventilator.

having a horizontal movement. These pistons are rectangular, in rectangular cylinders; each is 30 feet wide and 21½ feet high; each is carried on four rollers running on two rails. They do not touch the walls of the cylinder, but are as near a fit as can be without touching. Each cylinder has 168 suction and 196 delivery valves on each end, thus there are 728 valves in all; the fly-wheels make about seven revolutions a minute. The efficiency of the machine depends on the condition of the valves and the amount of leakage past the piston.[1]

Fig. 306 and 306*a* show the Struve ventilator. This is driven by a steam-engine, one cylinder 24 inches in diameter, 4 feet

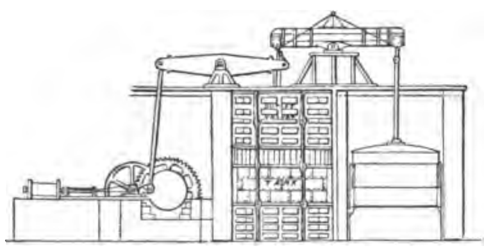


FIG. 306.—The Struve ventilator, Ynysdavid pit, Cwm Avon. Side elevation; chamber in section.

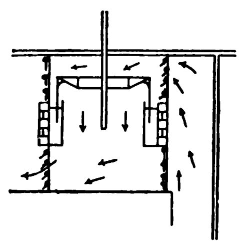


FIG. 306*a*.—Struve ventilator: section.

4 inch stroke. The speed is reduced by spur-gearing 4 to 1. Movement is given to two beams, each of which works a piston with vertical stroke 7 feet, and a diameter of 18 feet 3 inches, making 6½ double strokes per minute. The piston is

[1] The writer has never heard of any similar machine elsewhere.

made of sheet iron $\frac{1}{4}$ inch thick, in the form of a bell or gas-holder. The lower edge of this bell dips into a circular water-trough, and is never raised to the top of the water, so that no leakage of air can pass the piston. The piston is suspended by a rod working through a box in the cylinder cover, so that each piston is part of a double-acting pump. The useful effect of this machine depends to a great extent on the condition of the valves. It has been worked against a water-gauge of more than 5 inches.[1] There are 92 inlet and 92 outlet valves; each valve is 4 feet long, 14 inches deep, made of a light wooden frame with a zinc panel.

Rotary Air-pumps.—The reciprocating air-pumps above described have several serious objections. There are a number of valves to be kept in good order, and the speed must be slow, because, if a large piston is worked rapidly in alternate directions, the machinery may be soon destroyed (also the water-packing would be splashed out). To overcome these difficulties rotary air-pumps have been constructed.

One of these is Cooke's (see Fig. 307). Here the piston takes the form of a drum 22 feet in diameter and $11\frac{1}{2}$ feet wide. The drum is attached to an axle which does not pass through the centre, so that when the axle turns, the drum sweeps round like a paddle. It revolves in a chamber which is as near a fit to the sides and periphery of the drum as can be without actually touching. An orifice on one side of the chamber admits the air from the mine; another at the top allows the mine air to issue into the atmosphere. As the drum sweeps past the orifice from the mine, it pushes before it the air in the cylinder and forces it out at the upper orifice, at the same time sucking from the mine another cylinderful of air. In order that the mine air may be forced into the atmosphere, and not simply whirled round and round the chamber, a valve, x, is always in contact with the drum, or rather nearly touching. The mine air cannot pass this valve, and therefore is forced out of the upper orifice. When the drum sweeps by the

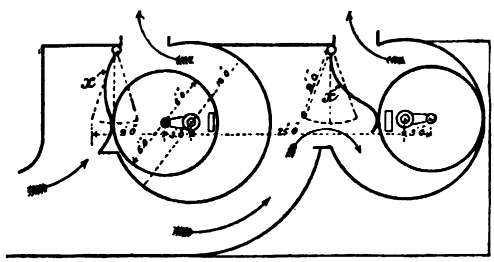


FIG. 307.—Cooke's ventilating apparatus.

[1] The writer is not aware of more than two of these machines in Great Britain.

valve at the greater radius from the axle, the valve is moved away by a system of levers outside the cylinder, so that the drum never actually touches the swinging valve, the latter following up the movements of the drum by means of the levers. The drum, being fixed eccentrically, would be heavy to lift were it not balanced; it is balanced, therefore, by another drum on the same axle or shaft revolving in another cylinder. By these two drums a regular current of air is extracted from the mine. An engine is attached to the axle or shaft carrying the drums. The efficiency of this fan depends on the accuracy with which it can be fitted in the cylinder without touching, and with which the swinging valve can be kept against the drum also without touching, and on the proportion of power absorbed by the friction of the revolving weights on the shaft bearings.

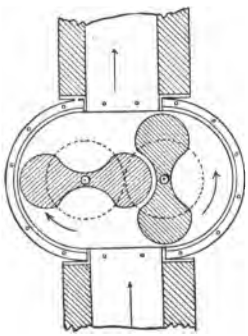


FIG. 308.—Root's small ventilator.

Another rotary pump is Root's (see Fig. 308). This figure represents the blower as used for blacksmiths' fires. Two drums shaped like the figure 8, with the longer axes at right angles, each carried on an axle through the centre; a spur-wheel is on each axle outside the casing, the teeth of which are engaged, and therefore each axle must revolve at precisely the same speed and in opposite directions. As each revolving piston passes the lower orifice, it drives before it and out at the upper orifice the air from the mine, which cannot return because the pistons are always nearly touching at the centre.

A large ventilator was erected at the Chilton mine, in the north of England (see Fig. 309). The length of the longer axis of each revolving drum or piston is 25 feet, and the length of the smaller axis 7 feet 6 inches. Each piston is 13 feet wide, and is covered with plate iron $\frac{1}{4}$ inch thick. The casing or cylinder in which they revolve is edged with cast iron, and wooden pieces adjusted by wedges diminish the clearance between the revolving pistons and the casing or cylinder to $\frac{1}{8}$ inch. The machine is driven by two cylinders 28 inches diameter and 4-feet stroke. On the crank-shafts are two bevelled wheels, gearing into other two wheels on the revolving shafts of the air-pistons.

The efficiency of this machine, as in the case of Cooke's, depends on the accuracy with which the pistons can be adjusted to the casing or cylinder, and on the amount of power absorbed in friction of the working parts.

Screw Ventilators.—As a current of air gives movement to a windmill, so a windmill, if driven by steam-power, would give movement to the air. Ventilators in shape like the sails of a windmill have been placed in a circular aperture in a wall, one side of which was connected with the upcast, the other side with the open air.

Sometimes, instead of a number of vanes as in the windmill, one vane only is used, but it is prolonged, so as to form a screw (see Fig. 310). This kind of ventilator is not adapted to mines, as it does not produce a sufficient difference in pressure, though it has been successfully employed for the ventilation of factories and other buildings, where the distance traversed by the air-current is very short.

There are many modifications of the rotary and screw type of machine, but they have not come into general use.

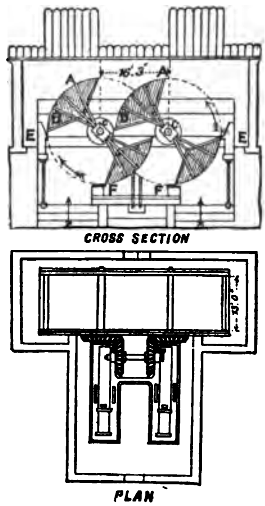


FIG. 309.—Root's large ventilator.

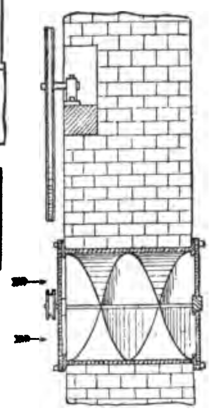


FIG. 310.—Screw ventilator.

Centrifugal Machines. — By far the greater number of mechanical ventilators are centrifugal machines, and there can be little doubt that they are at once the cheapest, simplest, and most efficient.

Centrifugal force is the force with which any body, revolving in a circle, tends to move away from the centre. The reason of this tendency is found in one of the laws of motion, that any body set in motion in any given direction will continue to move in that direction until some external force turns it another way, or stops it. Thus a wooden ball rolled along an alley will continue in the direction in which it is thrown until it comes in contact with a pin. A billiard-ball, if struck in the centre, continues in the direction given until it strikes another ball or the cushion. A rifle-bullet goes straight until the resistance of the air combined with the attraction of the earth causes it to fall to the ground. Therefore no loose body can move in a circle—it must be tied to the centre, as a young horse will trot round a circle when held with a cord, or the rim of a fly-wheel revolves in a circle, being held by the arms to the central boss, or a stone in a sling revolves in a circle until the sling is opened by loosening one string (when the stone immediately proceeds in a straight line).

In the same way, air can only revolve in a circle when it is enclosed in a case. If, therefore, a wheel with paddles revolves in a case open at the circumference, and with a central inlet (see Fig. 311) the air which is at the periphery of the fan immediately leaves the fan in straight lines.[1] The place previously occupied by the departing air is filled by air from the interior of the fan, entering through the central orifice. It has, however, been found that where there is a simple paddle-wheel, the place of the departing air is also supplied by atmospheric air, instead of entirely by air from the interior of the fan. This supply of atmospheric air is technically termed re-entry. The amount of this re-entry may be so considerable as to make the fan very inefficient.

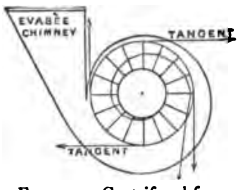


FIG. 311.—Centrifugal fan.

One of the most efficient centrifugal fans ever constructed is the old-fashioned winnowing-machine. In this the revolving paddles are enclosed all round, with the exception of three openings, one on each side at the centre for the admission of air, and one on the circumference for the escape of air; re-entry is thus prevented.

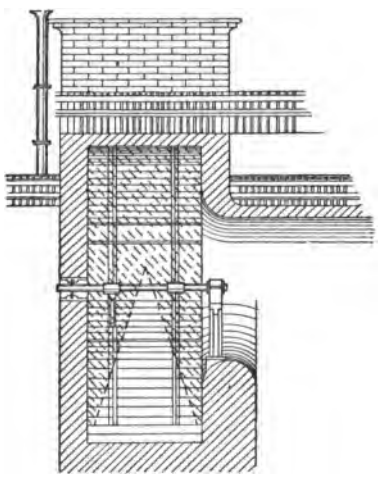


FIG. 312.—Guibal fan: front view of shutter. Dotted shading shows anti-vibrating shutter.

Guibal Fan.—The Guibal fan is an adaptation of the old winnowing-machine, for the ventilation of mines. Figs. 312 and 313 show the design. As usually constructed, it varies in diameter from 30 to 50 feet, and in width from 10 to 14 feet. The air enters on the side opposite to the engine, and is delivered into a vertical uptake. The orifice next the fan is contracted, 40 square feet being a suitable opening for

[1] These lines are tangents to the circle; for a straight line which passes the circumference of a circle, touching without cutting the circle, is a tangent. In the figure an external casing as in a Schiele fan is drawn, but for the illustration of this paragraph the reader must take no notice of this casing.

a fan 30 feet diameter and 10 feet wide, running at speeds of from 50 to 65 revolutions, the quantity of air passing being from 84.000 to 111,000 cubic feet per minute.[1]

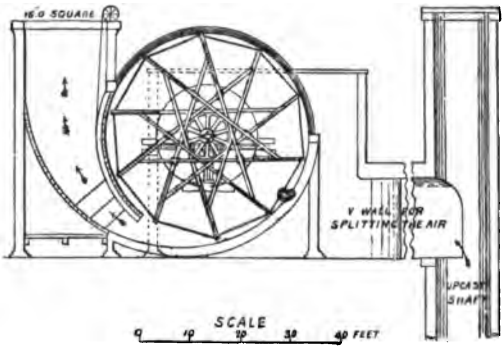


FIG. 313.—Leeds fan : longitudinal section.

The size of the orifice should be, in some degree, proportional to the quantity of air passing. The uptake is gradually enlarged to the top, making what is technically called an *évasée* delivery. The effect of this is that the speed of the current slackens as it approaches the top of the chimney. By some writers the high water-gauge obtained by this fan (see p. 226) is attributed to the shape of this uptake.

These fans have been very largely adopted. There is a good deal of vibration in them, sometimes leading to serious breakages; this vibration is much diminished by using the altered shutter, giving a larger and more gradual discharge (see Fig. 312). This figure

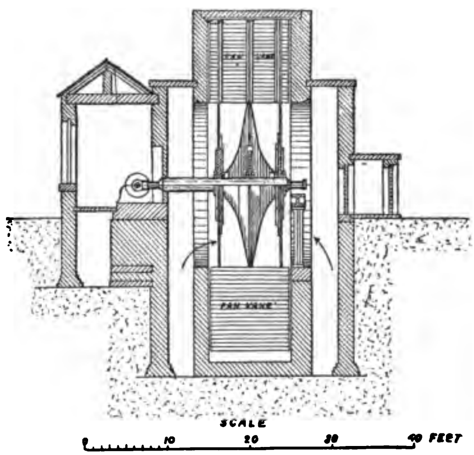


FIG. 314.—Leeds fan : transverse section.

[1] This is taken from a paper read by Mr. Robert Howe at the Chesterfield and Midland Counties Institution. Mr. Howe made a most careful and valuable series of experiments.

shows a V-shaped opening in the shutter indicated by dotted lines. Owing to their great weight, it is sometimes difficult to keep the bearings cool. The efficiency of some of them is given in Table X. (p. 229).

Leeds Fan.—The Leeds fan (see Figs. 313, 314, 315) is a modification of the Guibal, having two central orifices for the admission of air, and the width of the fan being rather less, a ventilator 40 feet in diameter being only 10 feet in width. These fans give satisfaction, and are subject to the same remarks as the Guibal as to vibration and weight.

Waddle Fan.—This fan (see Figs. 316 and 316*a*) is called an open running fan. It will be seen, on reference to the figures, that the fan is made of two circular iron plates or discs, one of which is flat and the other convex on the outside. The diameter of these machines varies from 20 to 50 feet. In the case of a 30-feet machine, the two plates are 3 feet 2 inches apart at the centre, and 1 foot 4 inches apart at the periphery, and are united by iron vanes or paddles.

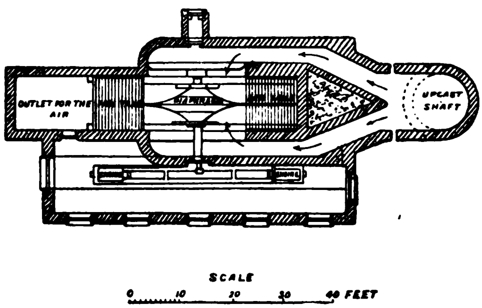
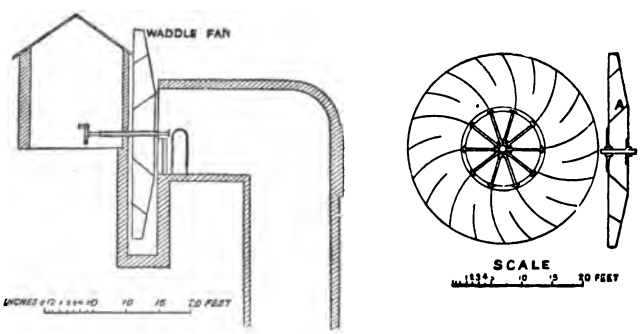


FIG. 315.—Leeds fan : ground plan.

There is one central orifice 12 feet in diameter, through which



FIGS. 316, 316*a*.—Waddle fan.

the mine air enters; as the fan revolves the air is expelled at the periphery, receiving a fresh supply from the centre. The central orifice has round it a wooden rim, which works close against the fixed rim in the wall, so that very little air can enter at the joint. This joint is also sometimes closed by a leather flap nailed to the stationary rim, which partially closes the joint. If properly made, very little leakage can take place. The fan is so shaped that the speed of the air passing through it is the same all the way along, and the width of the circumferential orifice is so narrow that the re-entry is not excessive. These fans are largely used, and give satisfaction.

Rammell Fan.—This fan (see Figs. 317 and 317*a*) is similar

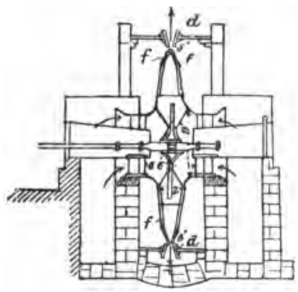


FIG. 317.—Rammell fan.

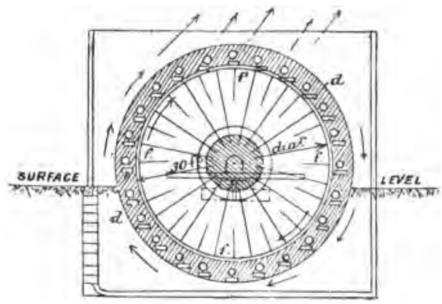


FIG. 317*a*.—Rammell fan. *d*, fixed adjutage; *f*, revolving fan.

to the Waddle, and has two central orifices for the admission of air, and the fan-shaft, being longer on this account, requires three bearings. The circumferential orifice is narrowed near the extremity to half the width, with the intention of reducing the chances of re-entry. Only a few of these fans have been erected; they give satisfaction.[1]

Gwynne Fan.—The Gwynne fan (see Fig. 318) is similar to a Rammell put inside a casing like a Guibal. This fan is 14 feet in diameter, and has two central orifices 5 feet wide; periphery, 10 inches wide; width at centre, 2 feet 4 inches; engine cylinder, 16 inches diameter; stroke, 16 inches; 150 revolutions per minute. This fan gives satisfaction.

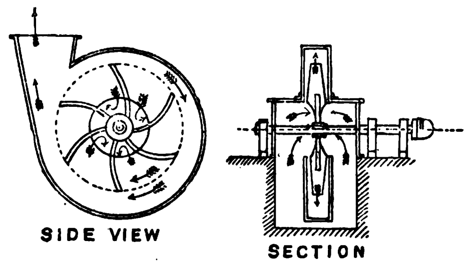


FIG. 318.—Gwynne fan, 14 feet diameter.

[1] The fan is also made without the adjutage *d*.

Bowlker and Watson Fan (see Figs. 319 and 319*a*).—This is like a small Guibal, 8 feet 6 inches in diameter, about 3 feet wide, two central inlets each 4 feet 8 inches diameter,

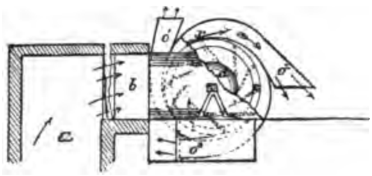


FIG. 319.—Bowlker and Watson's fan: elevation and part section.

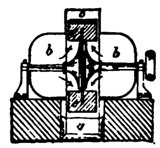


FIG. 319*a*.—Ditto: vertical transverse section.

three *évasée* deliveries. It seems to give good results.

Schiele.—Schiele's fan is a fan of small diameter, driven at a proportionately greater number of revolutions by means of a belt from the fly-wheel of the steam-engine (see Fig. 320). It is made in diameters varying from 5 feet up to 15 feet. It is in principle the same as the Guibal fan, with two central orifices for the admission of air; but the vanes are tapered towards the periphery, and the casing is eccentric, so that its *évasée* character is circular in form. A 12-feet Schiele had a width at the periphery of 2 feet 1 inch. A number of these fans are in use, and give satisfaction.

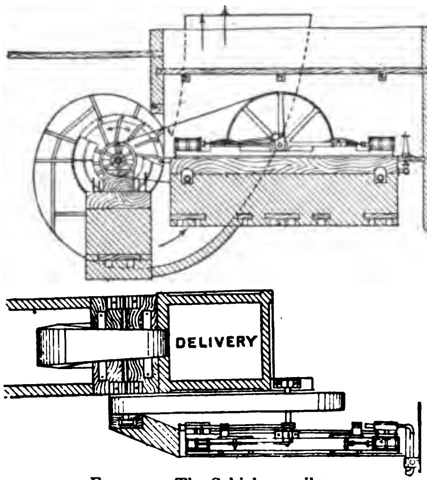


FIG. 320.—The Schiele ventilator.

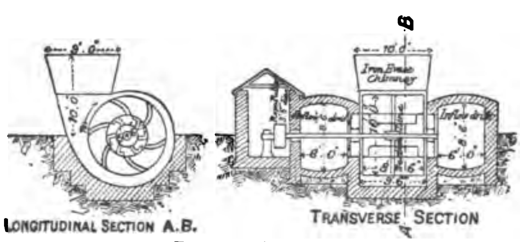


FIG. 321.—Capell fan.

Capell Fan (see Fig. 321).—This fan is very similar to the

Guibal, is sometimes made with one central orifice, and sometimes with two like a Leeds fan. It is of small diameter, varying from 5 to 15 feet. The vanes are wide, being 8 feet wide in a 10-feet fan. It is driven by a belt from the fly-wheel of the engine, or by cotton ropes, say six in number, running in grooved wheels. The speed of periphery is the same as in large fans, but the number of revolutions is of course greater. Mr. Capell attaches great importance to the shape of his vanes, but it may be a matter of controversy as to how far the peculiarities of shape affect the results. A number of these fans are now at work, and give satisfaction.

Medium Fan (see Fig. 322).—This fan was designed and erected by the writer of this treatise, and is named Medium because it is neither very large nor very small. It is similar to a Waddle fan revolving in a Guibal casing. It is 20 feet in diameter; the discs P, P are 2 feet 6 inches distant at the centre, and 9 inches distant at the periphery. The upper half of the casing in which the fan revolves is divided by a vertical partition DD on each side of the fan, so that the air in contact with the outside of the fan cannot revolve with it, but is deflected into the uptake. The air enters by the passage M through the orifice in one disc AA. The engine is attached to the crank K. All the dimensions are modified as required to suit each mine. The speciality of this fan consists in the careful proportionment of the casing, the proportions of the fan itself, and in the deflecting partition. The fan gives satisfaction.

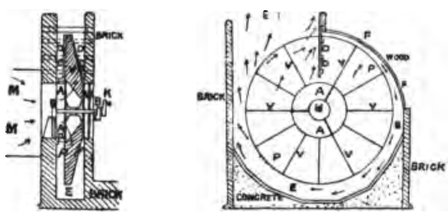


FIG. 322.—The Medium fan (patent [1]) : vertical sections.

There are many other ventilators, such as Lemielle's, Fabry's, Gunter's, Goffint's, etc.

Principles of Fan-construction.—The centrifugal ventilator is the best, because if carefully made very little power need be wasted. There are no valves, so that if strongly made it may last for generations, and any desired ventilating pressure can be obtained by its use.

The ventilating pressure produced by a fan varies as the square of the velocity of the fan-tips—that is to say, if the speed of the fan is doubled, the ventilating pressure will be quadrupled; if the speed of the fan is trebled, the ventilating pressure will be nine times; if the speed of the fan is quadrupled, the ventilating

[1] This was patented by Arnold Lupton

pressure will be sixteen times; so that to obtain any desired increase of ventilating pressure a comparatively small increase of speed only is required. The pressure that may be expected from a ventilating fan can be ascertained from the following rule:—

Let H = ventilating pressure in height of air-column.
V = speed of fan-tips (that is, periphery) in feet per second.
g = velocity given by gravity to a falling body in 1 second = 32 feet.

$$\text{Then } H = V^2 \div 2g, \text{ or } H = V^2 \div 64$$

Take an instance. Let V = 80 feet.

$$\text{Then } H = 80^2 \div 64 = 100$$

100 cubic feet of air at a temperature of 65° weigh 7·58 lbs., and that is the ventilating pressure produced per square foot. This divided by 5·2 gives 1·46 inch W.G.

If the speed of the fan is 160, then, applying the rule—

$$H = V^2 \div 64, \text{ or } \frac{160^2}{64} = 400$$

If the temperature of the air passing through the fan is 65°, then the ventilating pressure will be 30·32 lbs. per square foot, and this divided by 5·2, the weight of 1 inch of water over 1 square foot, gives 5·83 inches W.G.

In some fans, as, for instance, the Waddle and Rammell, the water-gauge is less than that due to this rule—say $\frac{9}{10}$ only of the theoretical water-gauge, as given by the above rule.

In the Schiele the water-gauge is often still less, say 0·66. In others, as in the Guibal, Leeds, Gwynne, and Medium, the water-gauge is more than the theoretical water-gauge, being equivalent to $\frac{11}{10}$, and up to $\frac{13}{10}$. But this is not always the case with these fans. The water-gauge falls when there is more than a moderate supply of air; the fall in the water-gauge is probably due to frictional resistance to the passage of a large current through the fan.

Many writers prefer to say $H = \frac{V^2}{g}$, or $H = \frac{V^2}{32}$. As the actual water-gauge can be ascertained by experiment, it does not matter which formula is used. If, however, $H = \frac{V^2}{32}$ is the formula used, then the actual W.G. of the Waddle or similar fans will be about $\frac{9}{20}$ of the calculated W.G., and the actual W.G. of the Guibal and similar fans will be about $\frac{11}{20}$ to $\frac{13}{20}$ of the calculated W.G. The real theoretical water-gauge can be calculated from the velocity of the air issuing from the fan-tips, but this velocity cannot easily be measured, whilst, on the other hand, the velocity of the fan-tips can be measured exactly.

It must not, however, be supposed that the efficiency of the fan necessarily corresponds to the water-gauge obtained at any given speed. Thus a fan that gives a low water-gauge may be very efficient, whilst the fan giving a high water-gauge may be comparatively inefficient.

The power of the engine driving a fan is absorbed, first, in giving velocity to the air inside the fan; secondly, in overcoming the friction of the piston, crank-pin, valves and journals of crank-shaft, and, in the case of a fan on the second motion, the friction of the journals of the fan, and in bending the belt: also, in the case of a Guibal fan, there is friction of the air against the sides of the casing; in the Waddle fan there is friction of the air outside the revolving plates, and there is also some re-entry.

The results actually obtained as recorded by various observers are given in Table X.

TABLE X.—FAN EXPERIMENTS.

Name of fan.	Dimensions.	Revolutions per minute.	Cubic feet of air per minute.	W.G. at fan in inches.	Useful effect per cent.
	Diam. Wide.				
Guibal ...	30 ft. × 10 ft.	60·0	104,299	2·95	65·24
,, ...	50 ft. × 12 ft.	40·98	108,422	3·3	40·0
Rammel ...	32 ft.	62·0	80,500	2·45	47·43
Waddle ...	30 ft.	60·0	89,160	1·7	58·22
,, ...	45 ft.	52·08	163,312	3·08	52·79
Schiele ...	12 ft. × 2 ft. 1 in.	180·0	157,176	1·91	46·12
Gwynne ...	14 ft.	101·0	17,150	1·7	57·37
Struvé ...	—	—	—	—	40·0
	2 pistons. Stroke.	2 strokes.			
,, ...	18 ft. 3 in. × 7 ft.	6·53	43,793	5·11	57·8
Lemielle ...	—	—	29,000	1·65	63·4
	Chamber. Height.				
,, ...	22 ft. 6 in. 32 ft.	9·9	47,307	1·37	23·4
	2 drums. Casing.				
Cooke ...	15 ft. 22 ft.	17·92	54,190	1·12	37·33
,, ...	—	26·0	101,308	1·12	64·0
	2 drums. Wide.				
Root	25 ft. × 13 ft.	18·0	101,696	5·0	64·19
,, ...	25 ft. × 13 ft.	16·71	89,772	3·29	47·84
Bowlker and Watson ...	8 ft. 6 in. × 3 ft.	150·0	19,860	1·27	57·6
Capell ...	10 ft. × 8 ft.	216 0	92,400	2·6	61·0
,, ..	12 ft. 6 in. × 5 ft. 8 in.	212·5	122,012	3·85	72·0
,, ..	12 ft. 6 in. × 5 ft. 8 in.	110·0	60,084	1·0	59·5
Medium ...	20 ft.	51·5	52,676	0·8	75·4
,, ...	20 ft.	60·0	51,392	0·95	65·5
Leeds ...	40 ft. × 10 ft.	50·0	254,500	2·6	60·0

Upon these results it may be observed that accuracy is got only with difficulty. To obtain it the following rules must be observed :—

The fan engine cylinder should be indicated at both ends simultaneously, and the indications should be continuous for, say, half an hour or more, because a slight fluctuation in velocity might lead to a considerable alteration in the diagram; secondly, the velocity should be taken with great accuracy by means of a stop-watch; thirdly, care must be taken that the water-gauge is protected from currents, and is so taken as to represent the pressure in the fan-drift, and not just part of the orifice of the fan.

The tunnel where the air is measured should be divided as nearly as possible into equal squares by means of strings or wires. The squares may vary from 1 square foot to 4 square feet in area; the smaller they are the more accurate the results obtained.

The anemometer should be held in each square for an equal period, say 10 seconds. After it has been held in each square, the distance recorded on the anemometer dial is the lineal measurement of the air that has passed the anemometer, and this divided by the time in minutes gives the average speed per minute.

This observation should be repeated several times, the engine being indicated, the number of revolutions counted, and the water-gauge observed, at the same time.

At least five observers are necessary if the measurements are to be taken with proper care.

The anemometer must be carefully tested at similar speeds to those of the air-currents.

The percentage of power absorbed by the friction of the bearings depends a good deal upon the weight of the machine, and the weight of the machine increases as the cube of the diameter.

If the diameter of one fan is 2, and that of another fan of the same kind is 4, then the weight will be as $2^3 : 4^3$, or as $8 : 64$. Therefore a 40-feet fan will be eight times as heavy as a 20-feet fan. If the diameter of the fan is doubled, the diameter of the shaft to which the fan is fastened will have to be more than doubled; if the same speed of periphery is maintained in each case, the speed at which the shaft revolves in the journal will be greater in the case of the large fan than in the case of the small one. Therefore, in the case of the large fan there will be not only eight times the weight, but this weight will be moving at a greater speed than in the case of the small fan, so that the shaft friction will be more than eight times as great if each journal is quite cool.

It is, however, often difficult to keep the journals of large fans cool, and it is not improbable that the friction of a 40-feet fan will be ten times that of a 20-feet fan for equal speeds of periphery, and as the work done depends on the speed of periphery, there will be ten times as much power wasted in the friction of the journals of the large fan. The good results obtained with small fans are thus easily accounted for.

It may be possible, however, to make fans too small, because when a large quantity of air has to be passed through it, the friction becomes excessive unless there is a good air-space. The small efficiency of some small fans is doubtless because a large part of the power is absorbed in the friction of the air passing through the fan itself; it would be mostly saved were the dimensions made a little larger.

Economy of Fans.—The amount of fuel required for one horse-power of steam varies from 1½ lb. up to say 30 lbs. of coal per horse-power per hour. The former figure is the best, practically, of the present day; the latter figure, absurdly wasteful as it is, is perhaps more common than some would suspect.

The usual consumption by common land engines, working continuously day and night, is from 5 lbs. up to 10 lbs. of fuel per horse-power per hour; but it is always possible, at a reasonable expense, to erect an engine suitable for driving a fan which would not consume more colliery slack than 4 lbs. per horse-power per hour.

Owing to the expense of maintaining the engines and boilers, 4 lbs. of fuel used in driving a fan will cost as much as 8 lbs. simply thrown on to a ventilating furnace. Therefore, if the fan gave only 50 per cent. of useful effect, and the engine consumed 4 lbs. of coal per horse-power per hour, 8 lbs. of coal would be required per horse-power in the air, and this would be equivalent in value to 16 lbs. of fuel in a ventilating furnace, which is the amount required per horse-power by a ventilating furnace at a depth of 1500 feet; so that at that depth and at greater depths the furnace would be more economical, while at less depths the fan would be more economical. If, however, the fan had an efficiency of 75 per cent., then 4 lbs. of fuel per horse-power engine would equal a consumption of 5·33 lbs. per horse-power in the air = cost of 10·66 lbs. burnt in a furnace, which is about the amount of fuel required for a furnace at a depth of 2400 feet. There is no reason why a fan should not give an efficiency of 75 per cent., or why an engine should not, using ordinary colliery slack, be driven with only 3 lbs. of fuel; and in this latter case the fan would be as economical as the furnace at a depth of 3000 feet.

In the fiery districts of the Midland Counties, Glamorganshire, and North and South Wales, the fan would always be preferred at a new colliery, irrespective of its economy.

Theory of Ventilation.—The following general rules, which are in accordance with the writings of the late J. J. Atkinson, if not exactly accurate, are sufficiently accurate for ordinary approximate calculations, and represent what may be called the orthodox theory :—

Rule I.—For the practical purposes of a miner, the friction of one particle of air against another need not be regarded. The resistance to an air-current arises from its contact with solid substances.

Rule II.—The resistance to an air-current, which is sometimes called "friction" and sometimes "drag," is proportional to the extent of solid substance with which it comes in contact.

Rule III.—The pressure required to impart velocity to an air-current varies as the square of the velocity given.

Rule IV.—The resistance to an air-current in the passages of a mine varies as the square of the velocity.

Rule V.—The resistance to an air-current in a passage varies inversely as the sectional area.

Rule VI.—The power required to overcome the resistance to an air-current in the passages of a given mine varies as the cube of the velocity.

In the following rules K represents the coefficient of friction; that is to say, the resistance of each square foot of surface of air-passage at a given velocity, say 1 foot a minute.

Rule VII.—The resistance measured in pounds per cubic foot of air-current = the coefficient of friction × the total extent of rubbing surface × the velocity squared, and divided by the sectional area of the air-road.

Let K = coefficient of friction.
A = sectional area of air-road in square feet.
V = velocity of air-current in feet per minute.
L = length of air-road in feet.
C = perimeter; that is, circumference if round, or four sides if square.
R = rubbing surface or superficial area of the air-passages in square feet = L × C.
P = ventilating pressure in pounds per square foot of A.

$$\text{Then } P = \frac{KRV^2}{A}$$

Let F.P. = foot-pounds of work done.
H.P. = horse-power.

Rule VIII.— $$\text{F.P.} = KRV^3$$ $$\text{H.P.} = \frac{KRV^3}{33{,}000}$$

For the purpose of comparing the friction of air-ways of different diameter, it is useful to bear in mind that the ventilating pressure varies inversely as the fifth power of the diameter; or, if D = the diameter, then—

Rule IX.— $$P \text{ varies as } \frac{1}{D^5}$$

The following rule shows how the coefficient of friction may be obtained:—

Rule X.— $$K = \frac{P \times A}{R \times V^2}$$

But for practically ascertaining the coefficient of friction of a mine in which the air-road has a varying section, it is necessary to calculate the dimensions of a hypothetical air-road, which shall have a resistance equal to those of the air-roads existing in the mine.

Let L' = the length of the hypothetical air-road.
A' = the area of the hypothetical air-road.
C' = the periphery of the hypothetical air-road, the dimensions of which may be fixed at will.
R = the rubbing surface in the actual road.
A = the sectional area of the actual road.

Rule XI.— $$L' = \frac{A'^3 \times R}{C' \times A^3}$$

Rule XII.— $$Q = \sqrt{\frac{PA}{KR}} \times A$$

A great variety of rules may be constructed out of the foregoing by any one who has mastered their principles.

Explanation and Illustration of Rules.—With regard to Rule I., it is sufficient to observe how freely the air moves when there are no solid obstructions—the wind blowing briskly on the top of a hill when in the valley below it is quite calm, or when there is a strong wind blowing over a plain, it is checked by passing through a grove of trees. Or in the case of a mine entered by a level from the hillside, the return air-course also being on the same level, and there being no vertical or incline shafts to cause a difference in pressure; in such a case, whilst a strong wind may blow against the opening of the mine, there will be very little wind in the interior, showing that the air-current is stopped by having to pass through a narrow passage.

Rule II. follows naturally from No. I. If it is contact with solid substances that impedes the current of air, then the more such contact there is the greater the impediment.

The practical value of this rule may be illustrated by referring to Fig. 323. Here is a circle and a square; each has the same area, but the square has the greater length of periphery—that is to say, the four sides of the square are longer than the circumference of the circle; therefore the air passing along the square road will be in contact with more surface, and will experience a proportionately greater resistance. Or, referring to Fig. 324, there is shown one road 10 feet square, the sectional area of which is 100 feet, and the periphery 40 feet. There are also four smaller roads, each 5 feet square, the sectional area of each of these roads 25 feet, and the periphery 20 feet; the

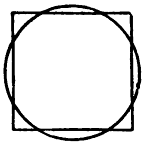


FIG. 323.—Circle and square.

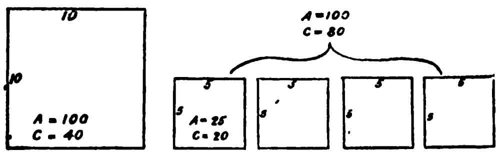


FIG. 324.—Square road and four small roads.

total area of the four roads is 100 feet, the total periphery 80 feet, which is twice the periphery of the single 10-feet road. Therefore the resistance to the air-current in passing through these four small roads will be twice the resistance to an equal current passing through the one large road.

Or, instead of four small roads, take the case (Fig. 325) of one road very wide and very low, say 37·4 feet wide, and 2·67 feet high, with a sectional area of 100 square feet. In this case the periphery is 80·14 feet, as compared with the 40-feet periphery of the 10-feet road, and the resistance is proportionately greater, the roads in each case being the same length. Or, to take another illustration, suppose one air-road to be 1000 feet long, and another air-road of precisely the same section to be 2000 feet long; there will be twice as much resistance in the long road as in the short one.

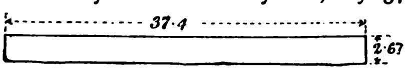


FIG. 325.—Narrow road and square road.

Rule III.—The demonstration of this rule, at first sight difficult to understand, is really exceedingly simple. In order to ascertain what amount of power is necessary to impart any velocity, it is desirable to make use of some uniform force. This force is generally ready to hand in the shape of gravity, or the attraction of the earth.

Every body attracts every other body in proportion to its mass. In proportion to the earth, all the small bodies on its surface are insignificant, and therefore everything, that is free to move, moves towards the centre of the earth. The atmosphere rests upon the surface of the earth, every building does the same. A stone dropped from the top of a building immediately seeks the centre of the earth, or, in other words, falls down. The nearer a body is to the centre of the earth, the greater the force with which it is attracted; but within small limits, such as a depth of a mile, the difference in the amount of attraction is insignificant, and for present purposes may be entirely disregarded. Therefore a body falling from the top of a high tower, or down a deep pit, is subjected to a uniform force—that is, the uniform attraction of the earth, which is always acting upon it, and from which it cannot escape. In a somewhat similar manner, the piston of a steam engine is influenced by a somewhat uniform force in the shape of the steam-pressure, and so long as the supply of steam is adequate and the area of the valves and ports is sufficient, the piston is subjected to a uniform force, no matter how fast the engine goes in trying to escape from this force. But with the best-regulated steam-engines it is impossible to maintain a really uniform force upon the piston at varying speeds, and therefore, for the purpose of scientific experiment, gravity is much more convenient.

It has been found by numerous experiments that if a body in a state of rest is allowed to fall during the time of 1 second, that it will fall a distance of 16 feet, if there is no material to obstruct it. If it were to fall in water it would not fall nearly so fast, and if it were to fall in air it would not fall quite so fast.

To ascertain the effect of gravity upon a falling body, it must fall in a vacuum—that is to say, in a chamber from which the air has been exhausted. The resistance of the air is a matter of common observation. A light body exposing a great deal of surface, like a feather, will float in the air, whilst a body of equal weight exposing very little surface, like a leaden pellet, will fall very quickly through the air; but in a vacuum a feather and a lump of lead will fall together equally fast.

It has also been ascertained, if a body falls during a space of 2 seconds, that it will fall a distance of 64 feet; and if it falls during a space of 3 seconds, that it will fall a distance of 144 feet; and if it falls during 4 seconds, that it will fall a distance of 256 feet. That is to say, the distance fallen is proportional to the square of the time during which the body falls.

Time in seconds.	Time squared.	Distance fallen in first second.	Total distance fallen.	Velocity attained.
1	1	16	16	32
2	4	16	64	64
3	9	16	144	96
4	16	16	256	128

From the above table it will be seen that the total distance fallen in any given time is equal to the square of the time in seconds multiplied by the distance fallen in the first second. The fifth column gives the velocity acquired in any given number of seconds, which the student can easily calculate for himself from the fourth column. Thus if a body starting from a state of rest falls 16 feet in one second, it is evident that 16 feet a second is the average speed of that second. But the speed at the beginning of the second was nothing; 16 is the average of nothing and 32, therefore 32 is the maximum speed, or the speed attained at the end of the first second. Again, at the end of the second second the total distance fallen is 64 feet; deducting from this the distance fallen at the end of the first second, 16 feet, the distance fallen during the second second is 48 feet, which is the average speed during the second second. But the speed at the beginning of this second was 32, and 48 is the average between 32 and 64, therefore the speed at the end of the second second must have been 64 feet. Again, the total distance fallen at the end of the third second is 144; 144 – 64 (the distance fallen at end of second second) = 80, which is the average speed, which is the mean between 64, the speed at the beginning of the second, and 96, the speed at the end of the second. The total distance fallen at the end of the fourth second is 256, and 256 – 144 = 112, which is the average speed and the mean between 96 and 128, which latter figure is therefore the speed attained at the end of the fourth second.

It thus appears that the speed attained is in direct proportion to the time, 32 feet being the speed attained at the end of the first second, and the number of seconds multiplied by 32 is the speed attained at the end of any given time.

It would be natural to inquire where a vacuum 256 feet in height could be found for the purpose of such an experiment. But the above laws can be demonstrated without the use of any such expensive apparatus as would be required to provide an exhausted hollow column 256 feet high.

Fig. 326[1] shows an incline down which a roller can run. This

[1] Designed by Professor Stroud, Yorkshire College.

roller consists of a solid metal wheel about $4\frac{3}{4}$ inches in diameter and $\frac{7}{16}$ inch thick, through the centre of which is a spindle about $\frac{3}{16}$ inch in diameter. The spindle is laid upon two thin iron rails on the incline, the wheel being free to revolve between the rails. When placed on the incline it naturally begins to roll down, and the friction of the spindle against the rails when placed at a gentle slope is such that it will not slide down, therefore it cannot go down the hill without causing the wheel to revolve. The wheel, being perfectly smooth, meets with very little resistance from the air as it revolves, and therefore nearly the whole effect of gravity is spent in causing the wheel to revolve. Had this wheel been simply dropped, the effect of gravity would have been expended in causing it to fall, and much more room would be required for such an experiment, whilst it would be much more difficult to observe. If, now, the wheel is allowed to roll down the incline at an inclination of say $2\frac{1}{2}°$, and a mark is made say at 1 foot down the incline, and another at 4 feet down the incline, it will be observed that the first foot is traversed in say $14\frac{3}{4}$ seconds, and the whole 4 feet is traversed in say $29\frac{1}{2}$ seconds. That is to say, that 4 feet is traversed in twice the time of 1 foot, which agrees with the rule previously given.

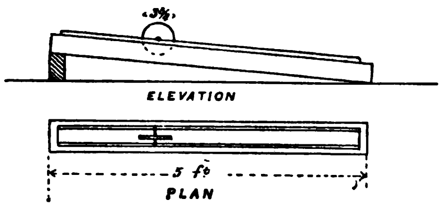


FIG. 326.—Inclined plane for gravity demonstration.

Accepting, therefore, this demonstration of the rule above given, we have an exact measurement of the power required to impart velocity. A velocity of 32 feet a second can be given to any body by raising it to the top of a tower 16 feet high, in falling from which it will acquire a velocity of 32 feet; or a velocity of 128 feet a second can be given to any body by raising it to the top of a tower 256 feet high, in falling from which (in a vacuum) it will acquire a velocity of 128 feet. Or, again, if we consider the case of the incline and have to roll the wheel up the hill, the gradient being uniform, it is evident that it will take four times as much power to move it the 4 feet as to move it 1 foot up the hill, and in rolling down the 4 feet it acquires twice the velocity that it does in rolling down the 1 foot. So that four times as much power has to be expended to give a velocity of twice. That is to say, the power required to impart velocity varies as the square of the velocity given.

This rule applies to every substance of which we have any knowledge, not merely to solids, but to liquids and gases. But in the case of blowing air through a mine, the velocity is not given by allowing the air to fall in a vacuum, but by the application of a steady pressure, and it may be useful to consider how far this is comparable to the effect of falling. If a cubic foot of air were to fall (*in vacuo*) 16 feet, it would acquire a velocity of 32, and the work done in giving that velocity would be equal to work done in lifting that cubic foot of air to the height of a column 16 feet. But it is evident to every mechanician that an equal amount of work would be done in lifting 16 cubic feet of air to a height of only 1 foot. If, therefore, we suppose a column of air 16 feet high, with a tap at the bottom of the column issuing into a vacuum; then the air would rush out with a velocity of 32, because, by the time the whole column had been lowered 1 foot as much work would have been used up as if 1 cubic foot had fallen 16 feet. And, again, if we let 1 cubic foot of air fall (*in vacuo*) 256 feet, it would have acquired a velocity of 128 feet, and the work done in giving that velocity would be equal to that in lifting 1 cubic foot 256 feet high; and, in the same way as in the previous instance, it is evident that an equal amount of work would be done in lifting 256 cubic feet 1 foot high. If, therefore, we imagine a column 256 feet high, with a tap at the bottom, the air would issue into a vacuum at a speed of 128 feet, because when the whole column had been lowered 1 foot as much work would have been used up as if 1 foot had been allowed to fall the whole 256 feet. And thus we see that the amount of work required to maintain a pressure equal to a given height of column during the time required to force a cubic foot of air from under that pressure, is equal to the work required to lift that cubic foot to the height of a column representing the pressure.

If, in the foregoing illustration, the student finds difficulty in imagining that any work has to be done in lifting air, which by his daily experience floats, let him substitute water for air in the column, and the analogy will hold good. It is thus evident that the pressure required to impart velocity varies as the square of the velocity given.

This can be easily proved by experiment. Take a glass jar open at the top, with an orifice near the bottom. Let this orifice be connected by a flexible tube $\frac{5}{8}$ inch bore with a glass tube about 1 foot long and $\frac{1}{4}$-inch bore laid horizontally; put two marks, one 6 inches above the other, on to the jar; fill the jar with water up to the upper mark, then hold the jar so that the water-level is 1 foot above the $\frac{1}{4}$-inch pipe, and note the time required for the water to run out to the lower mark. Then repeat the experiment, holding the jar so that the water-level is 4 feet above the pipe.

The time required will be half the time of the first experiment, showing that with four times the pressure the velocity is doubled.

Rule IV.—In considering this rule we leave the category of well-ascertained scientific facts, and enter upon an inquiry where the experience yet ascertained hardly justifies any unqualified statement. As before stated, Rule IV. is in accordance with the orthodox theory, and there is a great deal of reason and experience to show that it is approximate to the truth in many mines.

The friction, so called, of air in mines is by no means akin to ordinary mechanical friction. When a sledge is drawn across the ice, the horse or dog that drags it must apply a certain amount of force to overcome the friction, but that force will be the same no matter what the speed. In a similar manner, when a train is drawn along a level railway, the locomotive must exert a certain force to overcome the friction, but that force per mile of road is practically the same whether the speed is five miles or fifty miles per hour. In the same way with the fly-wheel of an engine, a certain amount of force is required per revolution to overcome the friction of the journals, but the friction per revolution will be the same whether the speed is ten or a hundred revolutions per minute as long as the journal is equally cool and well lubricated at both speeds. Therefore, to state that the friction of air through a passage increases as the square of the velocity, or increases at all as the velocity increases, is to show that a very different class of phenomena are under consideration. For this reason mining engineers often prefer the word "resistance" or "drag" to "friction" in dealing with air-currents.

By way of illustrating the reasonableness of Rule IV., the air-current might be likened to a waggon of coals. Suppose velocity to be given to this waggon of say 32 feet per second, but just when this speed is acquired the road suddenly turned at right angles; the waggon would run against the wall, and would come to a standstill. If, therefore, it had to continue on the way, the same power previously expended in giving it velocity would have now to be expended again, and so on at every right-angle bend in the road. If these bends were numerous in proportion to the length of the road, the power absorbed in reimparting velocity to this waggon after each stoppage would be so great that the power expended in friction of the wheel-axles would be so slight in comparison that it might be entirely neglected in the calculation, and it might be considered that the entire work done in moving that waggon along this road was the power expended in imparting and reimparting velocity, and we know that that would be proportional to the square of the velocity given. So that if, instead of a velocity of 32 feet per second, a velocity of only 16 feet per second were given, whilst the second velocity is half that of the first the energy required would only be $\frac{1}{4}$, or if a velocity of only 4 feet per second were given, which is $\frac{1}{8}$ of the first velocity, the energy required would be $\frac{1}{64}$.

When air travels through a mine it is continually passing round rapid turns, sometimes at right-angle bends, sometimes at a less or greater angle. Frequently the centre of the roadway is obstructed, and the air has to squeeze round the obstructions. More often the centre of the roadway is clear, but projections from the sides cause continual interruptions. Thus if the road is timbered, the air rushing along the sides and roof of the mine must be continually striking against the posts and bars. If it strikes at say an angle of 45°, it will glance off, but if it strikes directly it will bounce back, and its forward velocity will have to be reimparted by the absorption of power from the rest of the current. This impediment to the air-current is continual, and constitutes undoubtedly the greater part of the resistance that the air meets with in the mine.

It is only reasonable to conclude that if the resistance is due to these obstructions, it will be proportional to the number of these obstructions. Thus if in an otherwise smooth road there is one prop or bar for every yard, the resistance will be much greater than if there were only one prop in every hundred yards. Also if the roof and sides are uneven, forming a constant succession of cavities and projections, the air will be impeded just as if there were props and bars. If, on the other hand, the road is good and smooth, or is lined with smooth brickwork, the resistance will be much less.

A similar circumstance may be observed in water-channels. The water will flow along a smooth tube or over a well-laid invert with much greater velocity than along an irregular and stony channel.

Some interesting experiments have been made by Mr. Elwen.[1] He gives the resistance of air-courses of different descriptions, as shown in Table XI., p. 243.

Mr. Elwen is also of opinion that the resistance to an air-current in a mine varies as the square of the velocity, and has made some experiments which show that in some air-courses it is so. Ordinary approximate observations upon the water-gauge in a fan-drift, and upon the quantity of air passing through the mine, show that the *volume of air is approximately in proportion to the square root of the pressure;* or, as it is more commonly stated, that the *pressure is approximately in proportion to the square of the volume of air.*

It is, however, so difficult to measure the air, or to observe

[1] N. E. Inst.

the water-gauge with accuracy, and the range of pressure commonly observed is so slight, that it would be unjustifiable to set out a positive law for all pressures from these insufficient and generally rather discordant observations. Experiments especially made by the writer, with pressures up to 3 inches, agree with the rule. From experiments which he made with water in small tubes, Mr. Edgar C. Thrupp considers that the friction of water in small tubes is shown by the formula on p. 375. According to this rule, the increase of friction is less than it would be were it to increase as the square of the velocity.

The writer has made a number of experiments on the friction of air in small tubes, and finds that it is less than it would be if it increased as the square of the velocity, which seems to show that the resistance in smooth glass tubes varies according to a different rule to that of rough passages of a mine—a conclusion which would commend itself as being apparently very reasonable.

In conclusion, for the ordinary approximate calculations which are sufficient for ordinary mining operations, the rule is sufficiently accurate. Though, if it were desired to know the maximum quantity of air that might be passed through a long *smooth tunnel* at a given pressure, it would be advisable first to ascertain the rules that govern the flow of air in smooth tunnels, with a little more care than has perhaps yet been taken.

In practice Rule III. is not used, because the amount of power required to give velocity once to the air-current is so slight. Thus, supposing a final velocity of 16 feet per second, or 960 feet per minute, this would require a pressure equal to a column of 4 feet of air; or, turning this into a water-gauge, and taking water as being 800 times the weight of air, equal to $\frac{1}{200}$ part of a foot of water, or $\frac{12}{200}$ of an inch, which equals about $\frac{1}{17}$ of an inch of water-gauge—a pressure so slight that it can hardly be measured with an ordinary water-gauge.

Rule IV. is the most important rule, because it shows that it is impossible to increase the velocity of the air in a mine beyond a certain figure. Thus if, with a pressure of 1 inch of water-gauge, 50,000 cubic feet of air a minute pass through a mine, and it were necessary to increase the quantity to 100,000, a pressure of 4 inches of water would be required; or, if it were necessary to increase the volume to 150,000, a pressure of 9 inches would be required; and a pressure of 16 inches would be needed for a volume of 200,000 cubic feet, supposing that all the air-passages remained the same in each case. But pressures above 4 inches are unusual, and wasteful of power: therefore, in order to increase the volume of air in the mine, the air-roads should be enlarged.

Rule V. only requires a little consideration to be self-evident. Take the case of two air-ways of equal length; if the peripheries are equal, the rubbing surfaces in each road will be equal. Supposing the air-current in each case to have the same velocity, then the work to be done will be the same in each road. Let one road be 10 feet square (see Fig. 327), and the other be 15 feet wide and 5 feet high. The periphery of each of these roads is 40 feet, but the area of the 10-foot road is 100, and that of the 15-foot road is only 75. The rubbing surface and the velocity being the same in each case, there must be an equal ventilating pressure for the whole of the area of each road, and therefore the pressure per square foot must be greater for the smaller road, in order to make the total pressure equal in each case; so that if 1 inch of water-gauge were the pressure for the smaller road, 0·75 of an inch would be the pressure for the larger road. That is to say, the pressures are inversely proportional to the areas, other things being equal.

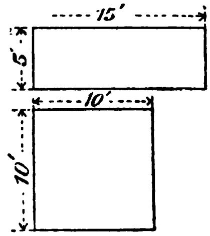


FIG. 327.—10-foot road and 15 × 5 road.

Rule VI. follows from the three rules, II., IV., and V. The mathematical terms, "squaring" and "cubing," will be easier to understand if we consider their derivation.

A square is a rectangle of which each side has the same length. If each side were divided into twelve parts, and lines ruled across from each division, forming it into little squares, the number of these squares would be 12 × 12 = 144, which is the square of 12.

If a cubical piece of wood were marked on each side of its upper surface with twelve divisions, and lines drawn across as before, there would be 144 squares on the surface. If one vertical side were also divided into twelve parts, and the cube was sawn across at each of its twelve divisions, each of these divisions would contain 144 small squares, which might be cut into 144 cubes; or there would be 1728 small cubes in the original cube. Therefore 1728 is the cube of 12, the cube of any number being that number multiplied by itself twice over, as 12 × 12 × 12 = 1728; 1728 is the number of cubic inches in a cubic foot. To speak of a number being cubed is merely a brief method of saying that it is multiplied by itself twice over. Another equally brief method is to speak of the number being raised to the third power; thus 12 to the third power is the same as 12 cubed, and it is written briefly thus: 12^3. In the same way, 12 squared is 12 to the second power, and it is written briefly thus: 12^2.

Thus the sixth rule means that the power required for ventilation is proportional to the velocity multiplied by itself twice. Thus if the air-current is 50,000 feet a minute, and the water-gauge pressure is 1 inch, the pressure in pounds is 5·2.

$$5{\cdot}2 \times 50{,}000 = 260{,}000 \text{ foot-pounds}$$

If the quantity is doubled, then the power, being proportional to the cube, will be—

$$50{,}000^3 : 100{,}000^3 :: 260{,}000 : x = 2{,}080{,}000$$

that is to say, the velocity being doubled, the power is increased by $2 \times 2 \times 2 = 8$. The reason of this is at once apparent; the quantity of air being doubled, twice as much power is necessary for this double quantity. But whilst the speed is doubled, the pressure is quadrupled; therefore four times as much power is required, so that we have 4 times × 2 times = 8 times.

Rule VII.—The coefficient of friction can only be ascertained by experience, and different results have been obtained by different observers, some of which are shown in Table XI.

TABLE XI.

	K for velocity of 1 foot in a minute.
The late J. J. Atkinson, in his well-known book, puts it at 0·0217 lb. for velocity of 1000 feet. If this is reduced to the coefficient for a velocity of 1 foot per minute, the figure would be in pounds per square foot	lbs. 0·00000002170
Devilez, a continental engineer, puts the figure at ...	0·00000000951
D. K. Clark (for railway tunnels)	0·00000000228
Arnold Lupton (the writer), longwall mine	0·00000000477
T. L. Elwen, for straight air-ways, even in section, without timber, in coal	0·000000002769
,, for straight air-ways, irregular section, without timber, in coal	0·000000003594
,, for straight air-ways, without timber, very jagged sides	0·000000004230
,, for straight air-ways, regular section, timber plentiful, in coal	0·000000004794
,, for shafts, timbered (buntons or brattice) ...	0·000000003686
,, for straight air-ways, very irregular section, with timber, in coal	0·000000005510
,, for straight air-ways in coal, irregular section, plenty of timber	0·000000005595
,, for air-ways round bord and pillar, face of workings	0·000000013685

It is evident, from these figures, that the coefficient varies with the nature of the surface of the roadway, and it seems probable that the average coefficient of a colliery would be about 0·000000005, which is also a convenient figure for calculations.

In the above table the coefficients of figures do not appear the same as in the works from which they are taken, because they are all reduced to a uniform velocity of 1 foot a minute, according to Rule IV.—that the pressure varies as the velocity². If, as in Mr. Elwen's case, the coefficient is given as 0·002769 for a velocity of 1000 feet per minute, the coefficient for a velocity of 1 foot a minute will be reduced by $1000^2 = 1{,}000{,}000$, and will therefore be written, 0·000000002769. The coefficient is often given in feet of air-column. Assuming that the temperature of the air is 52°, the barometric pressure 30 inches, then the weight of 1 cubic foot will be 0·078 lb. Then the coefficient above given may be turned into feet of air-column by the following rule-of-three sum :—

0·078 (lb.) : 0·000000002769 (lb.) : : 1 (foot of air-column) : 0·0000000355 (foot of air-column)

It may be useful to bear in mind that, at a temperature of 52° and a barometric pressure of 30 inches, a pressure of 1 lb. per square foot is equal to about 13 feet of air-column.

It is to be regretted that, while the coefficients above given differ so widely, there has not been some authoritative investigation of the question by one of the Institutes of Mining Engineers. The use of this coefficient is evident; without it we can only calculate the probable friction of air-roads by comparison with other air-roads, by means of Rules II., IV., V., and VI.; but the coefficient being once settled with approximate accuracy, it is unnecessary further to refer to other experience.

In future calculations, the coefficient will be assumed as 0·000000005 for a velocity of one foot a minute for the whole of the air-passages in an average colliery.

Rule VII. may be applied as follows :—

L = length of air-road.

C = periphery.

R = L × C = rubbing surface.

V = velocity in feet per minute.

Q = volume of air in cubic feet per minute.

K = coefficient of resistance per square foot of rubbing surface at a velocity of 1 foot per minute.

A = area in square feet.

In a given instance L = 8000

$$C = 26$$
$$A = 8 \times 5 = 40$$
$$R = L \times C = 8000 \times 26 = 208{,}000$$
$$V = 800 \text{ feet per minute}$$
$$Q = A \times V = 32{,}000$$
$$K = 0{\cdot}000000005$$

$$\text{then } P = \frac{KRV^2}{A} = \frac{0{\cdot}000000005 \times 208{,}000 \times 640{,}000}{40} = 16{\cdot}64 \text{ lbs.}$$

or, dividing by 5·2 to turn it into water-gauge, the pressure is 3·2 inches of water.

The accuracy of Rule VII. is evident if the previous rules are admitted, because P, the pressure per square foot, must be equal to the total resistance divided by the square feet of sectional area. This total resistance must equal the total area of rubbing surface R × the resistance of each square foot at a velocity of one foot K × the velocity^2.

Rule VIII.—On the same assumptions, Rule VIII. is also accurate, as is shown by the following reasoning :—

$$\text{Since } P = \frac{KRV^2}{A}$$

$$\therefore PA = KRV^2$$

Since F.P. = QP = quantity in cubic feet by pressure per square foot, and since Q = AV, then substituting AV for Q, we have—

$$F.P. = PAV$$

and substituting KRV^2 for PA, we have—

$$F.P. = KRV^2 \times V = KRV^3$$

Applying this rule to the road in the above example—

$$F.P. = 0{\cdot}000000005(K) \times 208{,}000(R) \times 512{,}000{,}000(V^3)$$
$$= 532{,}480$$

and dividing by 33,000 to turn it into horse-power, we get—

$$H.P. = 16{\cdot}13$$

The advantages of a large air-way are easily shown by applying this rule. Suppose that in the foregoing example the air-road had been only 5 × 4—

or A = 20
C = 18
R = 18 × 8000 = 144,000
Q = 32,000 as before
V = 1600 feet per minute

$$\text{then } F.P. = 0{\cdot}000000005 \times 144{,}000 \times 4096{,}000{,}000$$
$$= 2{,}949{,}120$$

or H.P. = 89·36, or nearly six times the horse-power required for the larger road.

It is evident that the velocity in the second instance is a great deal too high for practical purposes.

Rule IX.—The truth of this rule is also apparent (the same assumptions being made as before).

Thus P varies directly as the rubbing surface, and the rubbing surface varies directly as the diameter or side of a square, therefore P varies as D.

P also varies inversely as the area, and the area varies as the square of the diameter, therefore P varies as D^2.

P also varies directly as the velocity^2; the velocity also varies inversely as the area, or inversely as D^2; therefore P varies inversely as $D^2 \times D^2$, or as $\frac{1}{D^4}$.

Multiplying the above together, we have—

$$P \text{ varies as } \frac{D}{D \times D \times D \times D \times D \times D}, \text{ or as } \frac{1}{D^5}$$

Taking two shafts as examples, one 10 feet diameter and the other 12 feet diameter—

$$\text{For the 10-feet shaft, } \frac{1}{D^5} = 0{\cdot}00001$$

$$\text{for the 12-feet shaft, } \frac{1}{D^5} = 0{\cdot}000004$$

That is to say, for equal volumes of air the resistance of the 10-feet shaft is two and a half times that of the 12-feet shaft. This shows the great economy of large shafts for the purposes of ventilation.

Rule X.— $$K = \frac{P \times A}{R \times V^2}$$

This is evidently in accordance with Rule VII., and is applied as follows.

In any particular mine the ventilating pressure is measured by the water-gauge, from which the pressure, P, in pounds is then calculated.

A, the sectional area of the air-road, which is uniform throughout, R, the rubbing surface, and V, the velocity in feet per minute, are all ascertained by measurement.

It is evident that the coefficient of resistance per square foot of rubbing surface equals the total pressure, that is, P × A divided by the extent of rubbing surface in square feet, R, and also divided by V^2 to get the coefficient reduced to a velocity of 1 foot a minute. In applying this rule to any particular mine it is found that the air-road varies in section, and as the pressure required varies greatly with any change in velocity, it is necessary to ascertain the resistance due to each section of air-road.

The method of doing this is given by Mr. Thomas Fairley, in his excellent little book, "Ventilation of Mines," and the formula given in Rule XI. is taken from his book.

To explain the use of this formula, the simplest way is to take an illustration. Suppose the total length of the air-way to be observed is 4000 feet. This should be measured in say four hundred places. It will be found, however, that the sections so measured are in a great many places the same. For the sake of simplifying the example, we will assume that all the different measurements, each of 10 feet length, can be arranged under four sizes, each 1000 feet in length. (It is probable that in practice there would be many more sizes, and the lengths would be unequal.)

The four sizes are as follows:—

No. 1.	L = 1000	A = 30	C = 22	R = 22,000
No. 2.	L = 1000	A = 50	C = 30	R = 30,000
No. 3.	L = 1000	A = 40	C = 26	R = 26,000
No. 4.	L = 1000	A = 20	C = 18	R = 18,000

The total ventilating pressure for the whole mine = 1 inch water-gauge, or 5·2 lbs. Q = 15,000.

Let No. 1 be the standard section.

The problem is to find out what length of road of No. 2 section would have a frictional resistance equal to No. 1 section; also what length of road in each of No. 3 and 4 sections would have a resistance equal to No. 1 section. When this is ascertained we have a hypothetical road of uniform section throughout, to which Rule X. can be applied in order to ascertain the coefficient of friction.

L′ = length of road of standard section that would have a frictional resistance equal to that of the actual frictional resistance of the measured section.

The following table shows the result as calculated:—

		Feet.
No. 1.	L′ = L =	1000·0
No. 2.	L′ =	294·54
No. 3.	L′ =	498·57
No. 4.	L′ =	2761·36
		4554·47

The figures are ascertained as follows by Rule XI., in which

A′ = area of standard road = 30.

C′ = periphery of standard road = 22.

R = actual rubbing surface of the section measured.

A = actual area of the section measured.

L′ = length of road of standard size, the friction of which would be equal to the friction of the section as actually measured.

$$\text{No. 1. } L' = \frac{A'^3 \times R}{C' \times A^3} = 1000$$

$$\text{No. 2. } L' = \frac{30^3(A'^3) \times 30{,}000(R)}{22(C') \times 50^3(A^3)} = \frac{27{,}000 \times 30{,}000}{22 \times 125{,}000} = 294{\cdot}54$$

$$\text{No. 3. } L' = \frac{30^3 \times 26{,}000}{22 \times 64{,}000} = \frac{27{,}000 \times 26{,}000}{22 \times 64{,}000} = 498{\cdot}57$$

$$\text{No. 4. } L' = \frac{30^3 \times 18{,}000}{22 \times 8000} = 2761{\cdot}36$$

We thus find the hypothetical road is 4554·47 feet long, or rather longer than the actual length.

Applying Rule X., we ascertain the coefficient of friction.

The accuracy of Rule XI. (on the assumption made throughout, that P varies as the velocity²) is apparent by the following argument.

Comparing section 2 with section 1 (the standard section for the hypothetical road), it is evident that L′, the length of hypothetical road, with equal resistance to section 1, will be shorter than section 2, because the standard section is smaller, the resistance greater, and therefore a shorter length of that section will have a resistance equal to the longer length of section 2.

Considering first the reduction in length due to the increased velocity in the standard section, we have, calling V the actual velocity, and V′ the velocity in the hypothetical section, $V'^2 : V^2 :: L : L'$.

Also the ventilating pressure in the hypothetical section will be greater for equal lengths of road owing to the smaller area; therefore the length of hypothetical section will be again reduced in the following proportions:—$A : A' :: L : L'$. But owing to the less rubbing surface, due to the smaller perimeter of the standard section, the length of hypothetical road will be increased in proportion to the perimeters as follows:—$C' : C :: L : L'$.

Multiplying these together, we have—

$$\begin{array}{l} V'^2 : V^2 :: L : L' \\ A : A' \\ C' : C \end{array}$$

But since V′ is inversely as A′, and V is inversely as A, we have—

$$V'^2 : V^2 :: \frac{1^2}{A'^2} : \frac{1^2}{A^2}$$

$$\text{or } V'^2 : V^2 :: A^2 : A'^2$$

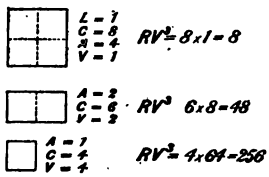
Then substituting A^2 for V'^2, and A'^2 for V^2, we have the following proportion:—

$$\begin{array}{l} A : A' :: L : L' \\ A^2 : A'^2 \\ C' : C \end{array}$$

$$\text{or } A^3C' : A'^3C :: L : L'$$
$$\text{or } L'A^3C' = LA'^3C = A'^3R \text{ (since } LC = R)$$
$$\therefore L' = \frac{A'^3R}{A^3C'}$$

Uniform Size of Air-road.—The foregoing rules are sufficient to prove that economical ventilation can only be obtained by having large air-roads throughout the mine. But it is not only necessary that the average section should be large, but that there should be no small or contracted passages, because the increase of friction due to a contracted passage is more than the reduction of friction due to an enlarged passage.

To take an instance from everyday life. One man takes a journey at 1*d.* a mile for 3 miles; total cost, 3*d.* Another man goes the first mile for 1*d.*, takes a cab the second mile, for which he pays 6*d.*, and walks the third mile to make up for his extravagance; total cost, 7*d.* In the same way, a contracted length of road may increase the resistance sixfold; whereas, if the next length of road is so large that the friction is practically nil, the previous increase of friction cannot be counteracted.



The following sketch (Fig. 328) shows two air-roads, the first one of uniform section, A = 2, the horse-power for ventilation = 18. The second air-road, of equal total length and larger average section, is in three lengths: one has an area A = 2, requiring six horse-power; the second has an area A = 1, requiring thirty-two horse-power (by Rule VIII.); the third has an area A = 4, requiring one horse-power; total horse-power for the uneven road, thirty-nine, or more than twice that of the road of uniform section and less average area, the average area of the uniform road being six as compared with seven, the average area of the irregular road.

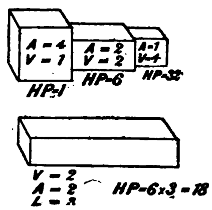


FIG. 328.—Air-roads of uniform and of uneven section.

If a train of waggons is left standing in the mine in a contracted portion of the air-way, there may be as much resistance to the wind passing through this part of the road as in the whole of the rest of the mine.

The conclusion to be drawn from these calculations is, that if there is a deficiency of ventilation in any mine or district of a mine, every contracted part of the air-way should be enlarged to the average or standard size. The brattice and air-pipes must be so fixed as not to obstruct the main current, and it is practically wasting money to make any part of the air-road much larger than the average.

Splitting and Coursing.—If the total quantity of air that could be passed through a mine were limited, as, say, by the dimensions of the reciprocating air-pump, it might be necessary that this air-current should pass through the mine in one undivided stream, making a long circuit in order that the current in each district should have sufficient velocity and momentum to clear out the gas from all the cavities and corners and high places in which it would lodge, and from which a current of slow velocity might not remove it. Such a method of ventilation is sometimes described as coursing.

If, on the other hand, the amount of air that passes through the mine is practically unlimited, as in the case of a mine ventilated by furnace or by a centrifugal fan (of which the dimensions are not too small), then it is unnecessary to course the air in one current. It may be split up into numerous currents, each taking one district.

This is the only safe way of ventilating a colliery, and is the most economical way. It is safe, because it introduces a supply of fresh air into each district, instead of trying to ventilate one part of the mine with air already fouled with gas from another part of the mine. It is economical, because the length traversed by each current of air is shorter, and therefore the ventilating pressure required is less.

The practical method of splitting the air-current by introducing air-crossings has been already explained and illustrated in Fig. 284. The total quantity of air passing through a mine may be enormously increased by splitting.

Assuming the case of two mines similar in every respect, the water-gauge or ventilating pressure being the same; but in one mine the air is coursed in one current, and in the other it is split up into numerous currents, say ten, each of these ten currents being, say, one-fourth the length of the course taken in the other mine. Then for equal velocities, the pressure required in the splits would be only one-fourth, but as the ventilating pressure is equal in each case, the speed along the shorter roads will be increased (Rule IV.) as the square root of the pressure, or as the $\sqrt{4} : \sqrt{1} :: 2 : 1$. Thus the speed in each of the ten roads is twice the speed attained in the other mine. Consequently, the area of each road being the same, the volume of air passed will be twenty times as great in the mine with ten splits as in the other mine with one long course.

The method of calculating the amount of air in any split is given in Rule XII.

$$Q = \sqrt{\frac{PA}{KR}} \times A$$

This rule is derived as follows :—

Q, the quantity of air in cubic feet per minute, = the velocity in feet per minute multiplied by the area of the air-road in square feet.

Or $Q = VA$, and according to Rule VII. $P = \frac{KRV^2}{A}$

$$\therefore V^2 = \frac{PA}{KR}, \text{ and } V = \sqrt{\frac{PA}{KR}}$$

$$VA = Q = \sqrt{\frac{PA}{KR}} \times A$$

In practice, to find the quantity of air passing along any given air-road, it will be best to apply this formula to each road separately; but for the purpose of illustrating the advantage of splitting, it may be simplified.

Fig. 329 shows a mine with four districts, through which the air is coursed in one stream, as shown by the dotted lines. Owing to the arrangement of the air-roads there shown (one which is hardly likely to occur in practice), it is possible to split up this air-course into four air-courses, as shown by the full lines, each about one-fourth the length of the original air-course. In this case the total length of air-road is the same before and after splitting, and therefore the rubbing surface is the same, the coefficient is also the same, and the ventilating pressure is maintained at the same water-gauge before and after splitting. So that P, K, and R can be neglected for the purpose of comparing the ventilation in each case, and—

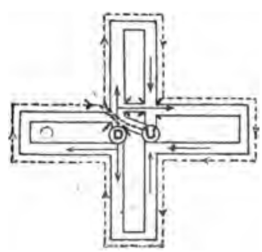


FIG. 329.—Method of splitting air. Dotted lines outside show air in one unsplit course; solid lines show four equal splits.

$$Q \propto \sqrt{A} \times A$$

As each of these air-roads is the same size, $A \propto$ the number of splits = N. Thus—

$$Q \propto \sqrt{N} \times N$$

Taking the above case, and assuming that Q = 10,000 in the first instance, when N = 1, after splitting N = 4. Then—

$$\sqrt{1} \times 1 : \sqrt{4} \times 4 :: 10{,}000 : 80{,}000.$$

It is evident that this is a much simpler example than usually occurs in practice. Suppose that in the instance above given the four splits had been of unequal length, everything else remaining the same; then, taking formula in Rule XII., we have P, A, C, and K constant for each split, L alone differing. Then the quantity of air in each road will vary inversely as the $\sqrt{\text{length}}$, written thus—

$$Q \propto \sqrt{\frac{1}{L}}$$

Let the air-course in the first mine be 10,000 feet in length, and be subsequently split into four courses of 1000, 2000, 3000, and 4000 feet each, or, as compared with the first air-course, 0·1 first split, 0·2 second split, 0·3 third split, 0·4 fourth split. Then the air passing along the first split as compared with that in the original course—

1st split $\sqrt{\frac{1}{0{\cdot}1}} = \sqrt{10} = 3{\cdot}162 \times 10{,}000$ cub. ft. (orig. quant.) $= 31{,}620$

2nd „ $\sqrt{\frac{1}{0{\cdot}2}} = \sqrt{5} = 2{\cdot}236 \times 10{,}000$ „ „ $= 22{,}360$

3rd „ $\sqrt{\frac{1}{0{\cdot}3}} = \sqrt{3{\cdot}333} = 1{\cdot}825 \times 10{,}000$ „ „ $= 18{,}250$

4th „ $\sqrt{\frac{1}{0{\cdot}4}} = \sqrt{2{\cdot}5} = 1{\cdot}5811 \times 10{,}000$ „ „ $= 15{,}811$

88,041

Additional Air-roads.—The effect of adding air-roads to a mine must not be confused with splitting. If a mine has one air-road, and another air-road is added of equal length and dimensions, and the ventilating pressure maintained, the total ventilation will be simply doubled; and for every additional air-road of equal length and area, an equal addition will be made to the amount of ventilation if the pressure is maintained.

Air-ways in Inclined Mines.—Where the mine is perfectly level, the temperature of the mine has no effect on the ventilation until the air arrives at the upcast shaft. Where, however, the mine is inclined, the temperature may have a considerable effect. For instance, if an incline is driven downhill, the fresh air from the downcast going down the incline and returning by another incline, the air gets warmed in the workings, and the inclines will be ventilated by a process of natural ventilation due to the temperature of the mine. If, on the other hand, the incline has been driven uphill, the cold air from the downcast going up this incline and returning down a similar incline, the air, getting heated in the workings, will be warmer in the return than in the intake, and this difference in temperature will work against the ventilation as much as in the other case it works in favour of the ventilation. There will be no natural ventilation in this case; the district will entirely depend on the pressure produced by the fan or ventilating furnace.

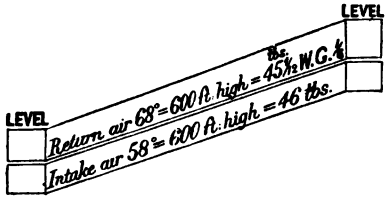


FIG. 330.—Incline air-roads, 2400 feet long, 1 in 4 W.G. more than ⅛ inch in *favour* for dip road; *against* for rise road.

To take a practical instance (Fig. 330). Suppose the incline to be 2000 yards in length, at a gradient of 10 per cent.; the total fall of the intake or rise of the return is 200 yards, and the ventilating pressure due to the temperature of the ground may be ascertained in the same way as for an upcast shaft 200 yards in depth. Let the average temperature of the intake incline be 62°, and the average temperature of the return incline 72°, then the ventilating pressure may be obtained by the rule given for furnaces (p. 207).

$$X = D - \frac{Dt'}{T'}$$

In this case D = 600, and $t' = 459 + 62 = 521$; $T' = 459 + 72 = 531$, and $X = 600 - \frac{600 \times 521}{531} = 12$; 12 feet of air-column at 62° = 0·17 inch W.G.

Or it may be found from the table of weights of air at different temperatures. One cubic foot of air at 62°, bar. 30 inches, weighs 0·07641 lb., and at 72° = 0·07497, or a difference of 0·00144 lb. for each cubic foot. This multiplied by 600 = 0·864 lb. Dividing this by 5·2, the water-gauge is found to be 0·166 inch, and this pressure is assisting the ventilation. If the incline had been driven uphill, this pressure would have been resisting the ventilation. If these inclines had been driven at a great distance from the shafts, where the ventilating pressure was very slight, this difference of 0·332 inch W.G. between the pressure required to ventilate the descending and ascending mines might make all the difference between having a good ventilation and an insufficient ventilation. If, on the other hand, the inclines had started from some place near to the shafts, where the available ventilating pressure was say 3 inches W.G., there would be practically no difference in the ventilation of the two districts.

Auxiliary Ventilation.—In some cases it is necessary to start air-roads of great length from a place which is already distant from the shaft, the ventilating pressure produced by the existing fan or furnace being required for the mine as it exists. But if a pair of roads are driven for a further distance of say a mile, it would be impossible to make the air travel that further distance without putting some regulator into the main air-course at the point where the new road branches, and thus obstructing the entire ventilation of the mine or the district from which these new roads are driven. Therefore, for the sake of ventilating these roads, it might be necessary to increase the ventilating pressure for the whole mine, requiring perhaps a considerable addition of outlay and working expenses. It may, however, be possible to ventilate these new roads by an auxiliary fan, which will produce the required ventilating pressure for this district alone. This fan may either force air into the new intake or exhaust it from the return.

This method is only practicable where steam-power is transmitted into the mine in some form, say by endless ropes, hydraulic pressure, compressed air, or electricity.

When compressed air is laid on, it is often used for ventilation. In continental mines, very small jets of it are taken into head-

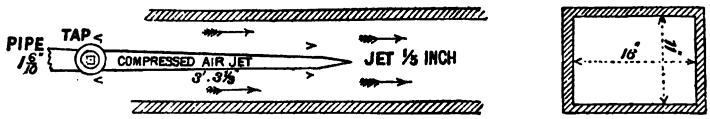


FIG. 331.—Compressed air-jet in pipes. *a*, wooden air-pipe.

ings. A better ventilation is, however, obtained by letting a jet of air blow through a pipe (see Fig. 331), thus inducing a considerable current; these small air-jets are, however, only for temporary makeshifts in special cases.

Barometer.—The barometer is used for measuring the pressure of the atmosphere. As usually constructed, it consists of a glass tube about a yard long, one end of which is sealed. It is filled with mercury, and the lower end placed in a cup containing mercury. The pressure of the atmosphere supports a column of mercury in the tube, the height of this column varying with the pressure. The pressure of the atmosphere at sea-level is equal to about 14·7 lbs. to the square inch, and this is equal to that exerted by a column of mercury 29·92 inches high. If the barometer falls, it indicates that the atmospheric pressure is less, and therefore gas is liable to be given off more freely from accumulations in goaves, headings, or fissures.

CHAPTER IX.

CHEMISTRY OF MINES: GASES, WATERS.

Gases.—The air consists of two gases, nitrogen and oxygen, which are not chemically combined, but are mixed up together mechanically, just as dry peas and beans might be mixed together in a bushel. The gases are both of them invisible, without taste or smell, and are very nearly the same weight, so that they do not separate one from another. The mixture is composed, according to Cavendish, of: oxygen, 20·833 volumes; nitrogen, 79·167 volumes; total, 100 volumes. When taken by weight, the mixture is composed of: oxygen, 23 lbs.; nitrogen, 77 lbs.; total, 100 lbs.

The specific gravity of air is 1. The symbol of oxygen is O; the atomic weight, 16; density, 16. It is the great supporter of life and of combustion; without it animals die, and a fire or candle will not burn. We could not live in pure oxygen, because it would be too exciting. If a red-hot iron poker is held in pure oxygen it will burn rapidly. The nitrogen in the air serves to dilute the oxygen and make it fit for ordinary use. Nitrogen, symbol N; atomic weight, 14; density, 14. It is a harmless gas, and does not support either life or fire, so that in an atmosphere of pure nitrogen human beings would die, and a light would not burn.

Fire-damp.—The noxious gases met with in mines are, first, fire-damp, called by chemists light carburetted hydrogen, or marsh-gas. Its symbol is CH_4; density, 8 (that is to say, it is half the weight of oxygen); the specific gravity is 0·5576. It consists of carbon and hydrogen chemically combined. It is colourless, tasteless, and without smell. It is often disengaged from decomposing matter in marshes, and might be lighted as it bubbles to the surface; it burns in the atmosphere, but does not give such a good light as ordinary illuminating gas. When mixed with air in proportions varying between 7 per cent. and 14 per cent. of gas, it is explosive; if there is 10 per cent. of gas, it explodes with great violence. It is found in most mines. No trace of it has been seen in the Forest of Dean, but in all the main coal-fields of the country it is abundant. It is given off not only by coal, but by strata of all kinds in the coal-measures. This is the gas which is meant when the word "gas" alone is used. Being lighter than air, it is found in the highest cavities, but in rapid currents it may get mixed up with the air. Whenever air and CH_4 are present together they are always more or less mixed, but the lighter gas is found in greater proportion in the higher part of the chamber containing it. In many mines it is given off so abundantly that it amounts to 2 per cent. of the return air. It is also often given off from coal after it has been brought to the surface, and any place where the coal is stored must be ventilated. If it is put into a ship, for instance, and the hold or bunker is not ventilated, there is likely to be an explosion if the place is entered with a lighted candle. It exists in coal under great pressure; it has been observed at a pressure of several hundred pounds per square inch; but very likely this measurement does not represent the greatest pressure at which it may exist. It blows out through innumerable small pores and crevices in the coal; owing to the great pressure at which it exists in the coal, its issue is not affected by any variations in the atmospheric pressure as shown by the barometer. Reservoirs of this gas in open places in the mine, whether in unventilated headings or in unventilated goaves, however, are subject to variations of volume corresponding to the variations in barometrical pressure. Thus if a cavity contains 300,000 feet of explosive mixture, and the barometer falls 1 inch, from 30 inches to 29 inches, the 300,000 cubic feet will expand $\frac{1}{30}$, that is to say, by 10,000 feet, and this volume of explosive mixture will issue from the cavity. There should be, however, in every part of the mine such good ventilation as will sweep away any expansions of gas, and there should, of course, as a general rule, be no large unventilated cavities. If 2 per cent. of this gas is in an atmosphere thickly impregnated with coal-dust, the mixture is explosive. This gas is not fit for breathing, and if there is a large percentage of it in any place, men entering it will be suffocated.

Black-damp.—Carbonic acid gas, called by chemists carbon dioxide. Symbol, CO_2; density, 22; specific gravity, 1·524. This is commonly found in mines, generally in shallow mines. It is colourless, odourless, invisible. Being 50 per cent. heavier than air, it is generally found on the floor and in cavities in the floor, and can be poured like water from a high place into a lower place. If breathed by human beings or animals they soon die, and flame is instantly extinguished.

The extinction of life and of flame are not due to any hurtful properties of the gas, but simply to the fact that it takes the place of the oxygen which is necessary for the support of life and flame. Thus a candle burning in a sealed chamber will be extinguished, not so much by the carbon dioxide produced by the combustion of the carbon of the candle and the oxygen of the air, as by the fact that the free oxygen, being reduced in amount, cannot get sufficiently rapid access to the flame to keep up the combustion.

If a candle is placed in air that has been expired from the human lungs, it is extinguished owing to the want of oxygen as much as owing to the presence of carbon dioxide.

Professor Frank Clowes, when at University College, Nottingham, made some valuable investigations on this subject.[1] The following table, taken from Dr. Clowes' paper, is instructive:—

EXPERIMENTS MADE BY PROFESSOR FRANK CLOWES.

Combustible substances burned.	Percentage composition of the residual atmosphere in which the flame was extinguished.			Proportions per cent. of O_2 and N_2 in which the flame is extinguished when introduced.	
	O_2.	N_2.	CO_2.	O_2.	N_2.
I.					
Alcohol, absolute ...	14·9	80·7	4·35	16·6	83·4
,, methylated ...	15·6	80·25	4·15	17·2	82·8
Paraffin lamp-oil	16·6	80·4	3·0	16·2	83·8
Colza and paraffin ...	16·4	80·5	3·1	16·4	83·6
Candle...	15·7	81·1	3·2	16·4	83·6
II.					
Hydrogen	5·5	94·5	—	6·3	93·7
Carbon monoxide ...	13·35	74·4	12·25	15·1	84·9
Methane ...	15·6	82·1	2·3	17·4	82·6
Ethylene (failed) ...	—	—	—	[13·2	86·8]
Coal gas	11·35	83·75	4·9	11·3	88·7
III.					
Expired air (average)...	16·15	79·9	3·95	—	—
Fresh air	20·9	79·06	0·04	—	—

The results obtained in the above table, parts I. and II., were obtained by placing a candle or lamp in a bell-jar, the pressure within the bell-jar being kept constant by an ingenious arrangement. It will be observed that the candle was extinguished when the percentage of carbon dioxide was 3·2, when, at the same time, the percentages of oxygen and nitrogen alone were 16·4 and 83·6. Part III. of the above table shows the percentage of carbon dioxide in air expired from the human lungs.

The following table shows the percentages of carbon dioxide required to extinguish a light when the mixture is obtained by

[1] See *Transactions*, Fed. Inst. of Mining Engineers, vol. vii. p. 419, and vol. ix. p. 376.

adding the carbon dioxide to the atmosphere. It is seen that the percentage of carbon dioxide required to extinguish a candle in this case is 14, and to extinguish a lamp burning colza oil 16, and to extinguish a flame of hydrogen 58.

EXPERIMENTS MADE BY PROFESSOR FRANK CLOWES.

Results obtained with naked flames.

Combustible substances burnt in the mixture.	Extinctive proportion of carbon d oxide added to the air.		
	Percentage of carbon dioxide added.	Percentage composition of the mixture.	
		Oxygen.	Nitrogen and carbon dioxide.
I.			
Candle	14	18·1	81·9
Colza and petroleum	16	17·6	82 4
Ordinary lamp paraffin	15	17·9	82·1
Alcohol, pure	14	18·1	81·9
,, methylated	13	18·3	81 7
II.			
Hydrogen...	58	8·8	91 2
Coal gas	33	14·1	85·9
Methane (fire-damp)	10	18·9	81·1
Carbonic oxide (white damp) ...	21	16·0	84·0
Ethylene	26	15·5	84·5

An atmosphere containing percentages of oxygen and carbon dioxide in which a candle will not burn may be breathed by human beings without inconvenience for some time. It has been found by Mr. J. R. Wilson[1] that rabbits can breathe with impunity air containing by admixture as much as 25 per cent. of carbon dioxide for an hour at least.

The presence of CO_2 can be ascertained by observing a candle or other light. In exploring wells it is common to lower a light down to see if there is any of this gas in it; where the light will burn animals can live. In small quantities the gas is not poisonous. It is found in the atmosphere to the extent of 4 volumes in 10,000 volumes of air, and is expired by animals in breathing.

Sulphuretted Hydrogen.—The chemical symbol is H_2S;

[1] Professor Clowes, *Transactions*, Fed. Inst. of Mining Engineers, vol. vii. p. 422; *American Journal of Pharmacy*, vol. lxv. No. 12.

its density is 17. It is a colourless gas, having an offensive smell like that of rotten eggs; it is very poisonous. It burns in air with a light blue flame, and will probably form an explosive mixture with air. It is occasionally, but very seldom, found in considerable quantities in mines. It is said to cause blindness if breathed. It is probably not uncommon in mines in minute quantities, which give a scent to unventilated places containing fire-damp.

Carbonic Oxide, generally called by chemists carbon monoxide.—Chemical symbol, CO; density, 14; specific gravity, 0·968. This gas is colourless and tasteless. It is combustible, burning with a blue flame; it is explosive when mixed with air in the proportion of 1 volume of carbon monoxide to $2\frac{1}{2}$ of air. It is exceedingly poisonous, a minute percentage producing fatal effects if breathed for a short time; it is the most dangerous gas ever found in a mine. Fortunately it is not, like fire-damp and carbon dioxide, a natural gas, but is only found as the result of imperfect combustion. It may be produced by the explosion of gunpowder and other blasting compositions, and is often present in smoke from a fire. The bright flame emitted from uncovered blast-furnaces is produced by the combustion of carbon monoxide. The density of carbon monoxide being about the same as that of air, it is equally likely to be found on the floor and near the roof.

Mixing of Gases.—The tendency of all gases is to mix together, and the only reason why fire-damp is found near the roof is because its tendency to rise (owing to its lightness) is greater than its tendency to mix; in the same way that carbon dioxide is found near the floor, because owing to its weight its tendency to sink is greater than its tendency to mix (see Figs. 332, 333). When the air is moving, the tendency to mix is increased. In a place containing a great deal of gas, whether the air is moving or stationary, fire-damp may be found on the floor as well as at the roof, air, being also at the roof, forming an explosive or inflammable mixture throughout the whole height of the place.

Gases also pass through most solid substances, such as a brick wall, and will pass very rapidly through a piece of unglazed porous porcelain. The lighter the gas the more rapidly it passes through porous substances; thus fire-damp will pass through more quickly than air, and air more quickly than carbon dioxide. If a stoppered bottle of porous porcelain containing air is put into the gas CH_4, the gas will pass rapidly into it, and the air will pass out at a less speed; consequently the bottle will get fuller, and the pressure inside will be greater.

The speed of diffusion of gases varies in the inverse ratio of the square roots of their densities; thus the proportionate densities of air, fire-damp, and carbon dioxide being respectively about 2, 1·1, and 3, the relative rates of diffusion will be about as $\frac{1}{\sqrt{2}}$, $\frac{1}{\sqrt{1 \cdot 1}}$, and $\frac{1}{\sqrt{3}}$, or as 0·71, 0·95, and 0·58, or as 71, 95, and 58.

Ansell's Indicator.—This principle was used by Mr. Ansell for the purpose of making a fire-damp indicator. He had a small cup covered with a plate of porous porcelain, the cup con-

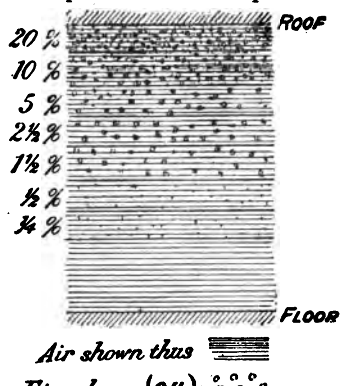


Fig. 332.—Mixing of air and fire-damp.

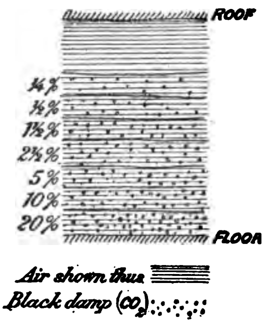


Fig. 333.—Mixing of air and black-damp.

tained mercury connected with a tube rising up like the tube of a barometer from the bottom of the cup. When the cup was placed in fire-damp, the gas entered the cup faster than the air escaped, thus causing an increased pressure in the cup, which caused the mercury to rise in the column; the rising column of mercury formed an electrical communication, which rang a bell. In another form of the instrument the porous diaphragm was put at the back of an aneroid barometer, and when held in fire-damp the barometer indicated the higher pressure. Neither of these instruments were of any practical use, however.

Liveing's Indicator.—A red-hot platinum wire will glow brilliantly if surrounded with an atmosphere containing fire-damp. This property has been adopted by Professor Liveing for the construction of a fire-damp indicator. Two platinum wires were placed side by side, one in an air-tight chamber, the other exposed to the atmosphere. A current of electricity passed through both by means of a small magnetic generator, caused both wires to glow; if the exposed wire were in an atmosphere containing fire-damp it glowed more brightly, and the percentage of fire-damp could be measured by the comparative degree of brilliancy. A compact photometer was added to the instrument. This instrument has also been found practically useless.

Spirit Flame.—The presence of small percentages of fire-damp can be detected by using a flame of some gas that gives very little light, such as that of spirits of wine, as noticed in the description of the Pieler lamp, Chapter XI.

Hydrogen Indicator.—The hydrogen flame has been adopted by Professor Clowes, of Nottingham, and its use described by him in a paper communicated to the Royal Society. For the working of this invention he has a small cylinder of compressed hydrogen, and this is connected by a small pipe with the interior of a safety-lamp (see Fig. 334). This figure shows a lamp as patented by Messrs. Clowes and Ashworth; it has, however, since been improved in many details, protected by a later patent, but the principle remains the same. H is the small steel reservoir containing hydrogen at a pressure of say 100 atmospheres; the escape of the gas can be regulated or stopped by a small screw plug, P. The reservoir can be quickly detached and carried in the breast pocket; the attachment or detachment can be effected in three or four seconds. B is the hydrogen pipe, at the top of which is a small burner, the flame from which is regulated by means of the screw tap to a height of 10 millimetres, or 0·4 inch. R is the oil-reservoir, W the ordinary wick, and G the pricker; T is the tube bringing the air to be tested, as in a Gray safety-lamp; A is the glass.

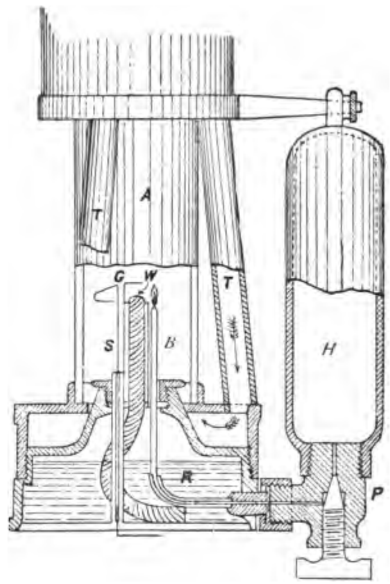


FIG. 334.—Hydrogen-gas detector. W, wick; B, hydrogen burner; R, oil-reservoir; P, regulating plug; A, glass; S, pricker for wick; G, gauge-wire; H, reservoir of compressed hydrogen; T, air-tubes.

When a test is to be made, the lamp is used as an ordinary safety-lamp, testing for gas with a full flame. If this reveals the presence of gas, there must be a large percentage, the amount of which may be judged by lowering the flame and observing the height of the cap in the ordinary manner. In case, however, the first test does not reveal the existence of fire-damp, the hydrogen cylinder is attached, and the hydrogen turned on, which lights itself at the oil flame; the oil flame is now put out by means of the pricker. The height of the hydrogen flame is then regulated, and if there is as much as $\frac{1}{5}$ per cent. of fire-damp a cap will be distinctly seen on a careful examination; with $\frac{1}{4}$ per cent. of gas, the cap is clear but faint, 17 millimetres high; if there is $\frac{1}{2}$ per cent., the cap is slightly longer, that is 18 millimetres, but is much brighter; with 1 per cent. the cap is 22 millimetres, and with 2 per cent. 31 millimetres, with 3 per cent. 52 millimetres. A millimetre is 0·04 inch. The above measurements are from experiments made with methane by Professor Clowes. The percentage is judged from the brightness as much as from the height of the cap. The oil flame is relighted at the hydrogen flame after the test. The hydrogen flame cannot be blown out or extinguished accidentally. In order to test for gas in any given place, the lamp is first used in the ordinary way as an oil or spirit-lamp; if this gives no indication, the hydrogen flame is next tried, and the oil flame extinguished. With this lamp as small a quantity of fire-damp as $\frac{1}{5}$ per cent. can be clearly detected. The utility of such a means of measuring small percentages of gas is evident, because if in the main return air-way of a mine there is an apparent percentage of say $\frac{1}{4}$ or $\frac{1}{2}$, and if some of the returns have a less percentage, then it is evident that in some parts of the mine where the gas is given off the percentage will be much greater, and may be, taking into consideration the dust which may also exist, in dangerous quantities, though undetected by the ordinary safety-lamp.

Analysis.—Chemical analysis is a very excellent way of ascertaining the existence of fire-damp in the return air-ways, but in order to have this analysis it is necessary to take samples of gas in bottles out of the mine to the laboratory. The gas may be got by taking down bottles filled with water, and then, in the place from which the sample of air has to be taken, holding the bottle upside down whilst the water falls out; the place of the water will then be taken by the gas in the vicinity. A very small percentage of gas can be readily ascertained by the method adopted by Professor Winkler, of Freiberg. Systematic analyses of the return air in the mines in fiery districts by some such means would no doubt be of great use, by giving colliery managers exact information as to the volumes of gases given off from their mines, and the cost would be very trifling.

Precautions.—The noxious gases above described are chiefly found in the coal-measures, but all mines require ventilation. The workmen, animals, and candles consume oxygen and give out carbonic acid, and not only carbonic acid, but other matter which is injurious to health. Good ventilation is absolutely required for these reasons. Wherever explosives are used the air is made very foul. After a heavy discharge in some drift, if the workmen return to the place before the current of fresh air has driven out the smoke, they will be all poisoned, and many disastrous accidents have occurred through omitting this precaution; but even if the smoke is not sufficiently thick to kill them, there may be enough to injure the health and shorten the life of the workmen. It is only a short-sighted economy which takes no notice of the requirements of health, simply because the workmen are not engaged for life, but only by the week at one particular mine. To take the most sordid view of the question, any condition of work which reduces the vigour and longevity of the workmen must increase the cost of labour.

Spontaneous Combustion.—When anything takes fire without the direct application of some pre-existing fire (such as a torch or match), it is called spontaneous combustion. This is very common in coal-mines, few districts of the country being free from this liability, though it is much more frequent in some localities than in others. Thus, for instance, the thick coal of South Staffordshire, the thick coal of South Warwickshire, and the main coal of West Leicestershire and South Derbyshire are continually subject to the form of spontaneous combustion called gob fires, whilst they occur less frequently in the other seams and districts of the country. The fire hardly ever originates in the solid mass of the seam, but in the gob, which contains slack which has not been thought marketable, and portions of the seam of coal which have not been extracted, such as overlying beds of coal and underlying beds of coal of inferior quality or difficult to get. It is found that the gob gets hot, so that the temperature of the air in mines only 600 feet in depth will be 70°, though there exists no fire in the mine. Sometimes this heat increases locally to such an extent that smoke and ultimately flame issue from the goaf; the flame, if not extinguished, will soon set fire to the solid pillars, and the whole mine be in a blaze. In some cases, however, pillars of coal in a main air-way will take fire. The cause of these fires is, perhaps, not entirely ascertained, and has been a matter of much controversy. It has often been stated to be due to the decomposition of iron pyrites contained in the coal or shale, which decomposition, it was alleged, might cause heat and ultimately fire. Another view has recently gained ground.[1] It is that coal has a strong affinity for oxygen, and when it is minutely subdivided in the shape of dust, it rapidly unites with

[1] Professor Vivian B. Lewes, on "Spontaneous Combustion:" British Association, 1891.

oxygen and so gets heated, and this action becomes abnormally rapid after a temperature of 100° Fahr. has been reached. If there is a great heap of it, the heat cannot escape, and the heat so gained facilitates the union of oxygen and carbon, so that the hotter the heap gets the more rapid is the combustion, until it actually reaches the burning point, somewhere between 700° and 900° Fahr.[1] The method of dealing with gob fires is treated under the heading of Methods of Working. Coal on the surface, if left in large heaps, is apt to take fire, especially if there is any artificial warmth near. For instance, a heap of slack thrown against the base of a chimney or the side of a boiler flue would rapidly take fire; in the same way, if a steam-pipe in the mine were covered over with slack, a fire would probably ensue very soon.

It is, perhaps, possible that in some cases the heat may be to some extent increased by the pressure of overlying strata on small pillars or lumps of coal, the crushing of which may generate heat in situations where the heat is not carried away by convection or radiation; some kinds of shale, also, are liable to spontaneous ignition.

Waters.—In sinking shafts water may be met with that has been stored up in the rocks for a great length of time, perhaps for ages, and which contains in solution mineral matter. Springs are often met with that are perpetually charged with mineral matter. As a general rule, the water that is pumped in a mine varies with the rainfall in the district, and only remains a period varying from a week to a month in the strata, on its way from the field or river through the strata to the pumps in the mine, and consequently the water as it reaches the pumps is not highly mineralized. In other mines the amount of water is very small, and has traversed the strata very slowly indeed, and has in consequence become highly charged. In some mines there is a great deal of salt in the water, and it is used for medicinal baths. In the carboniferous formation the water is generally impregnated with oxide of iron and with sulphides. In veins containing copper the water often contains a great deal of sulphate of copper; this sulphate of copper acts rapidly on iron, so that neither iron rails, pipes, nor pumps can be used, but wood and lead have to be substituted. In metalliferous mines it has happened not infrequently that large feeders of water have entered the mine, not on a direct downward course from the surface of the mine, but have descended by some fissures to a depth much greater than the mine, and then ascended by other fissures into the mine, and the water so entering the mine has acquired a temperature due to

[1] Coal, however, has been found inflammable at a temperature of only 350°, and some hydrocarbons inflame at a much lower temperature than coal.

the greater depth, in some cases in Cornwall amounting to 120°; this has made the working place so hot that it is hardly possible to continue the extraction of mineral. In some of the mines in the Comstock Lode, U.S.A., the temperature was so high, owing to hot springs, that it was necessary to cool the men by showers of cold water.

CHAPTER X.

COAL-DUST: DANGERS AND REMEDIES.

It is now recognized that coal-dust plays a very important part in colliery explosions, but it is only within the last ten years that this fact has become generally acknowledged. The Mines Regulation Act, 1887, is the first Act of Parliament dealing with mines in which there is any notice of coal-dust. Five and twenty years ago the idea that coal-dust was explosive would have been generally ridiculed; nevertheless, scientific men have always recognized that coal-dust might be inflamed by an explosion, and so increase the disastrous effects. Some French engineers and chemists seem to have been among the first to suspect that coal-dust itself might be the chief source of damage in a colliery explosion. As long ago as 1864, Monsieur Verpilleux compared a colliery explosion to the firing of a gun, suggesting that coal-dust represented the gunpowder and fire-damp the priming. In 1872 some mining engineers of St. Etienne made notes of a colliery explosion in which the chief explosive agent was proved to be coal-dust, and experiments made by them showed that a dense cloud of coal-dust would blaze with such rapidity as to be explosive. In 1875 Monsieur Vital, mining engineer, published an account of experiments he had made with coal-dust, proving that it could be instantly ignited with a flame, and would burn with violence. In 1876 Mr. William Galloway, mining engineer of Cardiff, sent a paper to the Royal Society on "The Influence of Coal-dust in Explosions." The object of this paper was to prove that coal-dust, and not fire-damp, was the principal agent in many disastrous colliery explosions. He has since contributed various papers, the effect of which is to show that coal-dust, and not fire-damp, is chiefly accountable for many of the great explosions. To people accustomed to dusty mines where candles and blazing lamps are freely used, gunpowder shots discharged, and furnaces employed for ventilation, it seems strange to assert that coal-dust is explosive, when the experience of a lifetime seems to controvert the statement. If, however, the subject is considered, there seems no reason why coal-dust should not explode, because a mixture of coal-dust and air has much of the same composition as a mixture of fire-damp and air, or as gunpowder. The composition of these substances may be stated as follows, by weight:—

	Gun-powder.	Fire-damp and air.	Coal-dust and air.
Carbon	10·88	5.5 { 4·9	6·45
Hydrogen ...	0·00	1·6	0·4
Oxygen ...	36·96	93·6 { 20·8	20·6
Nitrogen ...	10·30	72·7	72·2
Potassium ...	28·97	—	—
Sulphur ...	12·80	—	0·1
Ash ...	0·40	—	0·25

When gunpowder is burnt or exploded, the heat is produced by the union of carbon with the oxygen; when fire-damp is burnt or exploded, the heat is produced by the combination of the carbon and hydrogen with the oxygen; and when coal-dust is burnt, the heat is produced by the combination of carbon, hydrogen, and oxygen. The reason why fire-damp and air are explosive is that the nature of the gases is such that the particles of carburetted hydrogen can be mixed with the oxygen, so that exceedingly minute particles of one gas are in contact with particles of the other gas, and there is nothing to prevent instantaneous union between the two gases. In the case of gunpowder, the carbon and sulphur are ground up exceedingly fine, and then mixed with the saltpetre (which contains the oxygen), which is also ground very fine, so that each minute particle of carbon and sulphur has a particle of oxygen immediately touching it, so that there is no reason why all the particles should not burn at once, and this simultaneous burning produces the expansion which accompanies the explosion. If the particles of carbon and saltpetre, instead of being ground up small, were mixed in larger pieces, they would burn if a light was applied, but not so rapidly as to produce an explosion. In the same way, lumps of coal, when a light is applied, will burn if there is a supply of air, but will not explode, because there is no material quantity of oxygen in the interior of the lump; nor is it possible to get a sufficient quantity of oxygen into such immediate contact with a lump of coal that there could possibly be instantaneous combustion, and in the case of ordinary coarse coal-dust, the particles of carbon are not small enough to permit of instantaneous union with the surrounding oxygen. If, however, the dust is exceedingly fine, so that it will float in the air in dense clouds, then each minute particle of dust may be surrounded with sufficiently large proportion of oxygen to burn, and there is no reason why the flame should not extend with extreme rapidity from particle to particle so as to produce all the

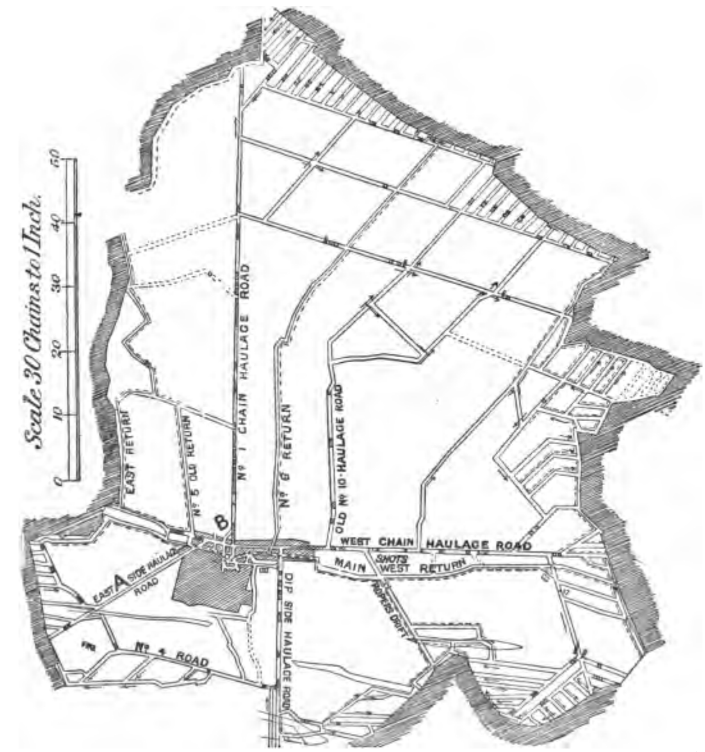


FIG. 335.—West Riding Colliery: plan. D, doors; S, sheets; X, overcast; ——, direction of wind; ═, intake; ≡, return; ▬▬, falls of roof; ═, horse-road.

effects which are known as an explosion. The experiments conducted by numerous mining engineers and chemists have demonstrated that if air is mixed with fire-damp in the proportion of 2 per cent. of fire-damp to 98 per cent. of air, this mixture is harmless; but that if into this mixture is put a dense cloud of coal-dust, and this is then ignited by a flame from a pistol, a violent explosion will ensue—proving beyond dispute that an atmosphere previously non-explosive becomes explosive when mixed with coal-dust. This is sufficient to place coal-dust amongst the dangerous elements of a mine, but there is evidence to show that coal-dust is much more dangerous than the above statement would convey. A commission appointed by the German Government made experiments in an artificial tunnel on the surface, with various kinds of coal-dust, with and without an admixture of fire-damp, and they succeeded, in at least one instance, in obtaining an explosion of dust when there was no coal-gas in the mixture. Explosions have also been recorded that have taken place in the hoppers of screens on the surface at collieries where the presence of fire-damp could not reasonably be expected, and which are generally accepted to be coal-dust explosions. But the evidence of several colliery explosions seems to establish the fact that coal-dust and air are terribly explosive without any admixture of fire-damp.

West Riding.—In October, 1886, there was an explosion at the West Riding Colliery, in Yorkshire; the verdict returned by

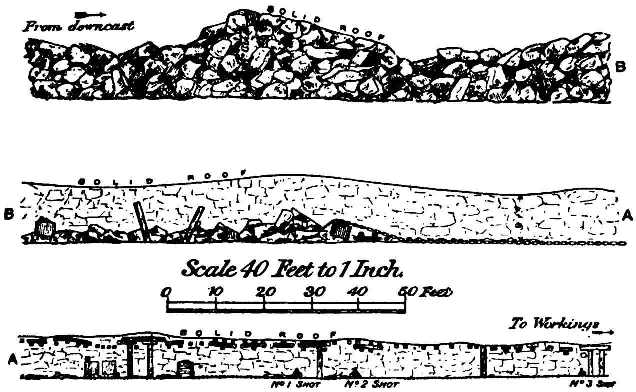


FIG. 336.—West Riding Colliery: section of west chain road, from origin of explosion at the shots going "out-bye" (towards the downcast shaft).

the jury, after a most careful inquiry, was to the effect that the explosion was an explosion of coal-dust ignited by a blown-out shot. The circumstances of the case may be understood by reference to the plan of the mine (see Fig. 335), and to the section of one of the roadways (see Fig. 336).[1] The colliery had the reputation of being very well managed and very well ventilated; it had been worked for twenty years with naked lights,

[1] See "Report of Royal Commission on Explosions of Coal-dust in Mines," evidence of Mr. W. E. Garforth.

but safety-lamps had been recently introduced as an extra precaution. The evidence showed that immediately before the explosion the mine was entirely free from fire-damp in any of the main roads or working places. The evidence also shows that the main haulage roads had in them a great quantity of coal-dust, some of which was of an exceedingly fine character, and when stirred up would float in the air in thick clouds. On the night of the explosion some men were at work making the west chain road wider, and to do this gunpowder was used; no gunpowder had been used on this road for many years. Two shots were fired; after the third shot was fired occurred the disastrous explosion. The suggestion is that the two first shots helped to shake up the dust, and that the third shot, having a long flame in consequence of being "blown out" or partially "blown out," ignited the compound of dust and air. The place of the explosion is marked on the plan, and the section shows the position of the shots. The section shows that there was no cavity where gas could lodge. It appears that the men who were engaged upon this work were killed by the explosion; but it does not appear to have been very violent at this place, as the timbers were not disturbed. At a short distance, however, going both "in-bye" and "out-bye," great signs of violence were apparent, and the signs of violence were stronger going "out-bye" towards the downcast shaft; on the way "out-bye" timbers were knocked down, and great masses of stone fell from the roof. About the shaft, survivors of the explosion observed a mass of flame; a great column of dust was thrown out of the top of the shaft. The effects of the explosion traversed all the main haulage roads, except one; but in no case was it observed that the explosion reached the working-face, and in no case were there any effects of an explosion in the return air-road. At the place where the shot was fired there was a very powerful ventilation—40,000 to 50,000 cubic feet per minute—and it was quite beyond experience or imagination to suppose that there could be fire-damp at this place; but even if there were a trace of fire-damp, it is impossible to suppose that there was any considerable quantity. If there had been any accumulations of fire-damp, they would in all probability be near the working-face, because here would be cavities in the goaf, not filled up, where gas might easily lodge, and here would be gases issuing from the coal, and if there was a blower of gas it would probably issue near the face, as, whilst there are many records of blowers, there are no records of any except those that have occurred near the face. But if there had been fire-damp issuing at the face or in the goaf, there would have been some evidences of explosion or combustion there; but, as a matter of fact, there were none. All the evidences of explosion and combustion were along the haulage roads, where there was fresh air from the downcast shaft and coal-dust. Near to the face there is not much dust, and what dust there is is of a larger kind; it is on the haulage roads that the dust is the most abundant and of the finest quality. It was above stated that the explosion extended into all the haulage roads but one; that one is the east chain road (A, Fig. 335) and between that road and the west chain road, where the explosion originated, are the stables. The ground near the stables (B) had been wetted by the water used in washing the horses; the explosion did not pass that place, whereas, if it had been an explosion of fire-damp, the water would have had little or no tendency to arrest the passage of the flame. But the conclusion seems to be irresistible: that this was an explosion of coal-dust without the presence of fire-damp.

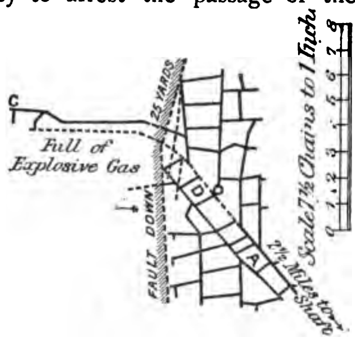


FIG. 337.—Whitehaven Colliery.

Whitehaven.—Some people have been under the impression that a comparatively small amount of fire-damp would produce the effects of a great colliery explosion. This, however, is not the case. Messrs. W. N. and J. B. Atkinson, inspectors of mines, in their exceedingly valuable book called "Explosions in Coal-mines"—a work which cannot be too carefully studied by all who wish to have a thorough comprehension of this question—give an instance of a fire-damp explosion at the Whitehaven Colliery (see Fig. 337). In this case, a length of say 380 yards (C to D) of heading contained an explosive mixture amounting to about 32,800 cubic feet,[1] which was fired, and the effects of the explosion were little felt or noticed beyond the point A. The mine was damp, so that there was no coal-dust to carry forward the explosion, and as soon as the exploded gases emerged from the heading and found room for expansion their force was dissipated, and this the more quickly because the heat which causes the gases to expand would be quickly absorbed by the cold surfaces with which the gases would come in contact.

West Stanley.—Another instance of coal-dust explosion given by Messrs. Atkinson is the West Stanley Colliery. Here was a coal yielding a considerable amount of gas, worked pillar-

[1] Messrs. W. N. and J. B. Atkinson, "Explosions in Coal-mines."

and-stall. Gunpowder was used, and a shot X, shown on the plan (Fig. 338), is supposed to havè ignited some fire-damp, and as a consequence the explosion traversed the whole of the mine, with the exception of two districts. The effects of fire were seen in both intake and return roads. In the intake, the fire travelled back to the downcast shaft; in the return, it did not reach the upcast shaft, but stopped at a place in the return road which was damp. The flame of the explosion, which had passed along the intake air-road to the downcast shaft, turned to the right hand into the south district, but did not pass the shaft into the north

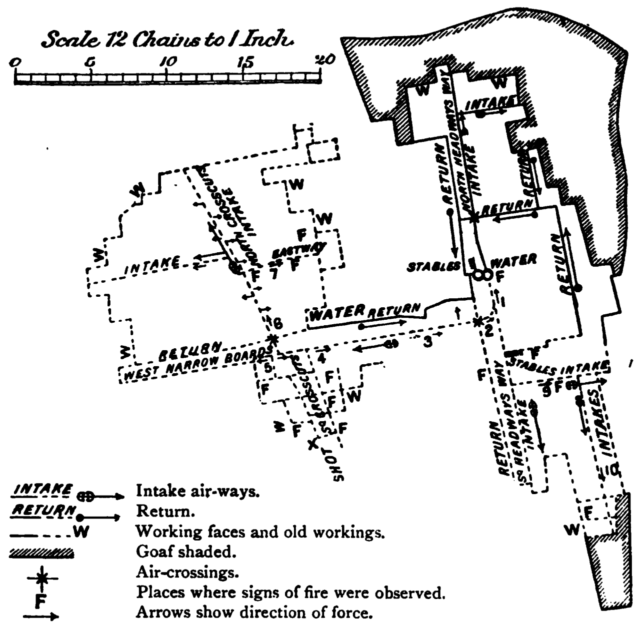


FIG. 338.—Plan of West Stanley Colliery.

district. The downcast shaft was wet, and the immediate vicinity of this shaft was damp. It thus appears that the explosion was stopped in two places by some moisture. If the explosion had been in gas, it would have passed over the moisture; but the moisture was sufficient to lay the dust, and therefore, if dust were the explosive, there was an absence of explosive material in this place, consequently the explosion ceased. According to the evidence, it is improbable that there was such an amount of gas in the workings as could be detected by ordinary examination with an ordinary oil safety-lamp, but it is probable that there might have been a small percentage in the atmosphere, and that this was reinforced by dust stirred up by the explosion, which was originated by the shot igniting, perhaps, some small accumulation of explosive gases. This is a case, not of a pure coal-dust explosion, but of an explosion rendered disastrous by coal-dust.

Elemore Colliery.—Perhaps one of the most extraordinary coal-dust explosions was that which occurred at the Elemore Colliery in Durham, in December, 1886. The evidence shows that the explosion originated in a main intake haulage road within 200 yards of the downcast shaft. It was an old colliery; the road had been made for many years, and there was a current of 37,000 cubic feet a minute of fresh air passing the place where it is said the explosion originated. On the night of the explosion some miners were at work enlarging the road at the place marked + on the plan (Fig. 339) and also on the section (Fig. 340). Several shots were fired by one shift of men, who were relieved by a second shift, who fired another shot, after which the mine exploded. The evidence seems to show that it was impossible for the explosion to have originated in any other part of the mine. The flame was seen to rush into the downcast shaft, and go down the shaft and enter a seam of coal on a lower level and pass along the intake haulage road, and go still further down the shaft and enter another seam on a still lower level. Everywhere the flame traversed *intake haulage roads* for a total length of 3500 yards. There was no explosion in the return air-roads; there was no explosion in the working places. The intake haulage roads were dry and dusty, and it is inconceivable that there could have been fire-damp in these intake roads and no fire-damp in the working places and returns. The conclusion is irresistible that this was an explosion of coal-dust.

Clay Cross.—To go further south, in the year 1882 an explosion occurred at Clay Cross, in Derbyshire, and there is little doubt that this was an explosion of coal-dust.

Pen-y-Graig.—At the Naval Colliery, Pen-y-graig, in Glamorganshire, an explosion occurred in the year 1880. The explosion traversed all the roads and working places in this pit, with the exception of one heading, which was damp. In this case the coal yielded a great deal of fire-damp, and it is probable that there would be a small percentage, not exceeding 2 per cent., in the return air-roads. The pit was exceedingly well ventilated; this was easy to arrange, because the downcast was at the extreme dip and the upcast at the extreme rise of the mine. It would be strange that the effects of heat and violence should have been strong in the intake air-roads if it had been a fire-damp explosion, and that one heading alone, which happened to be

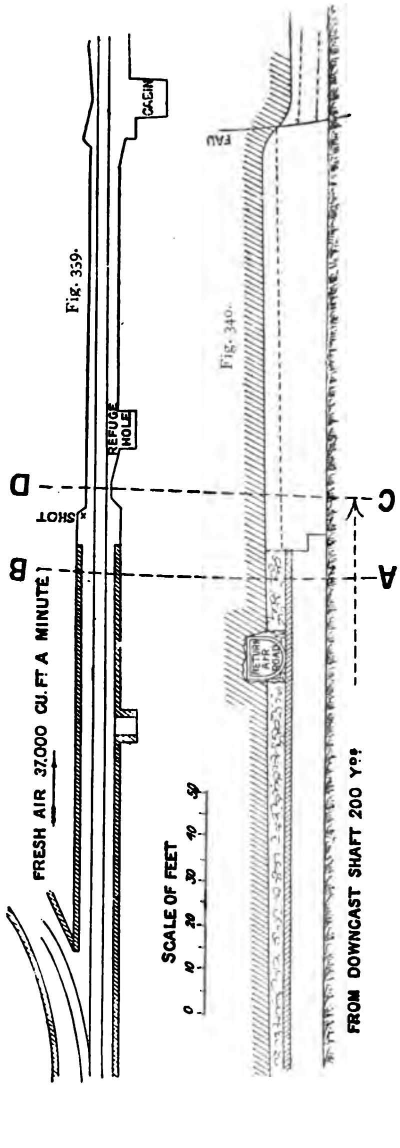


FIG. 339.—Elemore Colliery. Plan of part of intake, engine plane, where explosion originated at the shot.

FIG. 340.—Longitudinal section of the road shown in Fig. 339.

FIG. 341.—Transverse sections of the road shown in Figs. 339 and 340 at A B and C D.

the only damp place, should be the only place that was free from the effects of the explosion unless the explosion were one of coal-dust.

Lessons.—It would be possible to multiply instances of explosions in which coal-dust has played the chief part, and the more the subject is studied the more the evidence seems to show that a large proportion of the great explosions that have occurred during the last thirty years have been chiefly coal-dust explosions, though no doubt in many cases fire-damp has played an important part either in originating or in assisting the explosion. The lesson to be drawn from this fact is that coal-dust must be treated with care as if it were gunpowder; no shot must be fired in any dusty places until the dust has been laid by water. The intake air-road, which has for so long been considered the safest place where naked lights might be waved about and gunpowder shots exploded, must now be reckoned among the most dangerous places, not simply because it is an intake air-road, but because the intake air-road is generally the haulage road. Whichever is the road along which the coal is hauled will be the dusty road and the dangerous road. To remove this danger it is necessary to remove the dust. Coal-dust may be avoided or mitigated by various expedients. One way of not having coal-dust on the haulage road is to be careful that no coal is allowed to fall from the coal-waggons as they pass along, for that reason the sides and bottoms of the trucks must be closely jointed and they must not be overloaded, or, if coal does fall off, it must be picked up and removed. But there may be coal from the sides and floor of the road which might form into dust under the influence of the traffic over it. Experience shows that this dust can be laid and made solid if it is damped.

Dryness of Mines.—The intake air-ways of a coal-mine are generally very dry, for the following reasons: A mine in winter-time is much warmer than the air on the surface, and in summer-time a mine 200 yards deep is as warm as the air on the surface in the daytime, and warmer than the night air; a mine 400 yards deep is warmer than the air on the surface in the daytime in the warmest weather in England. When air is warmed its capacity for absorbing moisture is increased, and when it is cooled its capacity is diminished. Thus if in summer-time air passes into a shallow mine, say 50 yards deep, the roof of the mine will be damp with water, deposited by the incoming air cooled by contact with the colder ground. But in the same mine in winter-time, the incoming air being colder than the mine, it will be warmed, and the moisture in the roads will be sucked up by the air, and the roads will be perfectly dry. In a deep mine the air is always warmed as it enters the mine, and consequently its capacity for absorbing moisture is increased; it therefore sucks up all the moisture from the surface of the roads along which it passes until it is saturated. When the air reaches the newly cut coal at the face, it absorbs the damp from this and becomes less dry. It has also by that time generally reached the temperature of the mine. Such air now proceeds along the return air-roads; it does not receive any further increase of temperature, and its capacity for drying the roads is less than when it first entered the mine, but the main roads along which it has swept are as dry as tinder.

Methods of laying the Dust.—In order to lay the dust, one method is by taking a common water-cart along the road, at the rear of which is a pipe pricked with small holes, by which the roads are watered as in a street (see Fig. 342). Another method

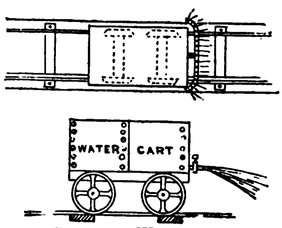


FIG. 342.—Water-cart.

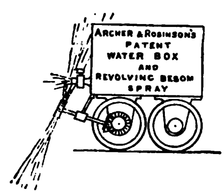


FIG 343.—Water-cart: revolving brush.

(see Fig. 343) is to have a water-cart, and at the rear is a revolving brush driven by gearing from the axles, which spatters water all round, so wetting the roof and sides as well as the floor. Another method is shown in Fig. 344. In this case a small force-pump is worked

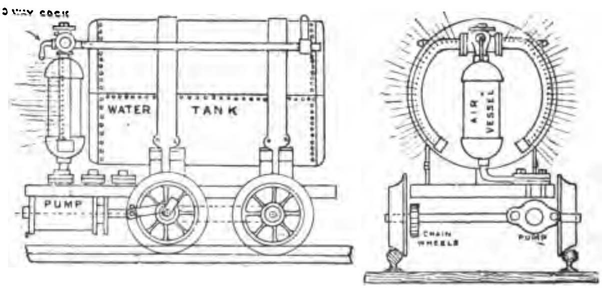


FIG. 344.—Kirkhouse and Lewis's patent automatic watering-tank.

by the carriage axle, and pumps water through a circular pipe in small jets, which strike the roof and sides and floor. Another method is to take water down the shaft and along the intake roads in pipes, say a 2-inch pipe in the shaft, and 1½-inch on the level;

a stand-pipe ½-inch bore every 25 yards, and at the top of the stand-pipe a small jet (see Fig. 345). The water in the pipes has a very high pressure, say from 80 to 250 lbs. on the square inch. The jet is exceedingly small, as small as can possibly be made; it is turned in the direction of the air-current. The water issuing from this very fine jet takes the form of a very fine spray or mist, which is carried along by the air-current, strikes against the roof, sides, and floor, and lays the dust. This method has been devised and carried out by Mr. John James Thomas, at Ynishir Colliery, Rhonddafach valley, Glamorganshire. Mr. Thomas says (and he is supported by other experienced colliery managers who have adopted this method) that the effect of this spray on the main road is to reduce the temperature and to lay the dust, and there are no ill effects; the roads are not wet, as by a water-cart, and therefore the floor of the mine is not injured, as would be the case if water were put on it. The tendency to dry rot in the timber, so common in dry mines, is also arrested, and the timber lasts well. By means of taps on some of the water-pipes, water can be got for the ponies, which is a great advantage.

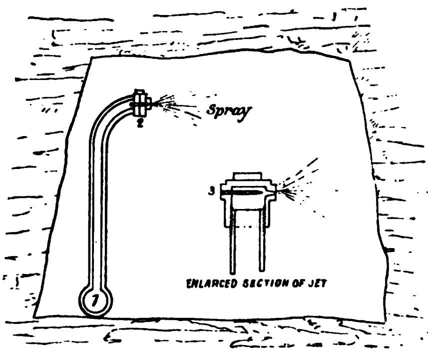
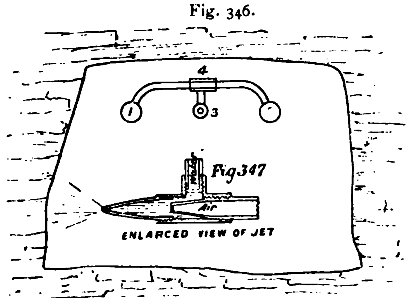


FIG. 345.—High-pressure water-spray. 1, 1½-inch water main; 2, jet for spraying water, regulated by screw 3.

Fig. 346.



FIGS. 346, 347.—Compressed-air water-spray. 1, main air-pipe; 2, main water-pipe; 3, mixed jet spray; 4, junction of pipes.

At the Dowlais steam coal-pits a system of spray-jets is used in which compressed air forces the water out through a small jet, making a very fine spray (see Figs. 346, 347). The system of water spray-jets has also been adopted throughout the Llwynypia collieries; at Harris's Navigation (Fig. 348) they have 15 miles of water-pipes; and at other places a similar method is used with great success. In fact, the system is becoming common in the steam-coal collieries of Glamorganshire. Another system has been tried at the Pochin pits, new Tredegar Colliery, by Mr. Henry Stratton. This colliery is 315 yards deep; the temperature of the return air-course was 63° to 64°. Mr. Stratton takes the exhaust steam from the fan into the downcast shaft, thereby raising the temperature of the downcast air to 63° before it enters the mine, and saturating it with moisture from the steam, so that it does not extract moisture from the mine. Therefore there is no dust made, and the pit is much pleasanter, safer, and healthier. This system is ingenious, but it is not always possible to carry it out in practice effectually, because sometimes in frosty weather the amount of steam necessary to saturate the air is such as to cause a mist at the pit-bottom, which is inconvenient. In some places the dust is bodily removed from the pit by shovelling it into waggons. The reader is referred to the Mines Regulation Act, 1887, for the rules now enforced relating to coal-dust.

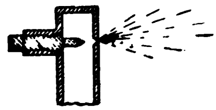


FIG. 348 — Spray-jet enlarged (Harris).

With regard to the injury producible by a coal-dust explosion, the experience obtained in mines shows that the violence is extreme, tossing heavy weights about, smashing waggons, timbers, doors, overcasts, blowing cages up the shaft, and otherwise destroying whatever comes in its way. But in order to arrive at some theoretical estimate of the injury to be expected, it must be borne in mind that, no matter how much coal-dust there may be, only so much of it can explode as enters into combination with the oxygen. But the only oxygen available for an explosion in a mine is that contained in the air.[1] Wherever there is a great deal of coal-dust, there is much more of it than can be burnt by the oxygen in the air in that locality, and it is therefore safe to assume that there is always a sufficiency of coal-dust in "every coal-dusty road," and that the force of the explosion may be measured by the amount of oxygen. In this way it has been calculated in the case of the Seaham Colliery, which exploded in the year 1880, that the amount of oxygen in the roads showing signs of fire would, if exploded with coal-dust, exert as much energy as say 50,000 lbs. of gunpowder. The length of roads traversed by the explosion amounts to 7500 yards in the aggregate, which equals 22,500 feet. If there was an average sectional area of 50 square

[1] There may be a good deal of oxygen in the coal-dust, possibly three times the volume of the dust, but this will only be about $\frac{1}{800}$ part of that required for combustion. It may, however, assist in the explosion.

feet, there would be 50 cubic feet of air in each length of 1 foot, weighing say 3·8 lbs., of which about 0·87 lb. would be oxygen. One pound of gunpowder contains about 40 per cent. of oxygen, so that 0·87 lb. of oxygen is contained in 2·17 lbs. of gunpowder; so that, on the assumption that the oxygen causes as much expansion with coal-dust as with gunpowder, the effect would be as $22{,}500 \times 2{\cdot}17 = 48{,}825$ lbs. of gunpowder. This rough calculation is based on a great many assumptions, and is put forward as an illustration, and not as a statement of facts.

CHAPTER XI.

SAFETY-LAMPS.

In the last century coal-mining was a comparatively small industry, and the methods of dealing with noxious gases but imperfectly understood; ventilation was frequently defective, so that accumulations of gas were very common. Experience having taught the miner the danger to be encountered on entering the mine with a candle, it became a practice in some parts to send in a man of bold and resolute character, wrapped in damp clothes, with a light at the end of a pole, with which he could set fire to accumulations of explosive gases, and thus render the place comparatively safe for the collier. These men received the appropriate name of "firemen," a title which is still given to the deputy who searches for gas at the present day. Other methods of working in mines containing explosive gas were attempted, such as the faint illumination given by the phosphorescence from the scales of fishes' skins, or the sparks produced by holding a piece of flint against a rapidly rotating wheel of steel.

Several ingenious lamps were tried by Humboldt, Dr. Clanny, and George Stephenson, but the first thoroughly good safety-lamp was invented by Sir Humphrey Davy in 1817. The Davy lamp now in use only differs in comparatively unimportant details from that constructed seventy-five years ago. The principle of the lamp is as follows: The flame is due to the union of gas—produced by heating the oil—with oxygen from the surrounding atmosphere. This union will not take place unless the temperature of the uniting gases is 1500° Fahr. If a gas-flame is brought in contact with a cool material, the flame is extinguished, and the gases pass away unburnt. This may be tested by holding a wax match against a cold poker, when the size of the flame will be greatly diminished. If, instead of a thick bar of iron, a number of small wires, woven into a gauze, are placed over the flame, the gases will pass through, but not the flame, because the gases are cooled by contact with the gauze. That the gas has passed through the gauze may be easily proved by lighting it with a match at the upper side (see Fig. 349). That the stoppage of the flame is due to the cooling action may be proved by holding one part of the gauze in the hottest part of the flame until it is red hot, when the flame will pass through, and the gas above will burn.

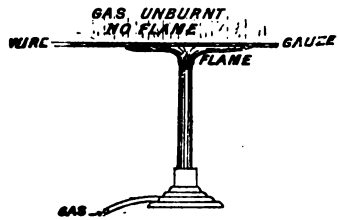


FIG. 349. Gas and gauze.

Davy Lamp.—The Davy lamp (see Fig. 350) consists, therefore, of an ordinary oil-lamp covered with a cylinder of gauze about $1\frac{1}{2}$ inch in diameter and 7 inches in length. The top of the gauze is covered over with a second cap. The wires are about $\frac{1}{50}$ of an inch in diameter, and there are 28 parallel wires in an inch, forming 784 apertures in a square inch; the area of opening is thus less than $\frac{1}{5}$ of the area of the gauze. Through these openings the air can freely enter so as to produce a good flame; but if gases should enter with the air, and, taking fire, fill the interior of the lamp with flame, as soon as this flame touches the gauze it is instantly cooled, and therefore cannot pass outside. Should a current of burning gas be kept against the gauze it will soon get hot, and the flame will pass, and the stronger the current of burning gas the more rapid will the heating of the gauze become; so that, whilst the Davy lamp is safe in a place where there is no current for a short time, it is unsafe in a current or if held in inflammable gas for more than a few minutes. The time required for the gauze to cool the flame is evidently only a very little time; but if the wires were made thinner without being placed closer together, the flame would have a less distance to travel in passing through the apertures, and less time would be occupied; and if the wires were too thin there would not be sufficient time for cooling, or, if the flame were projected with great velocity against the gauze, there might not be sufficient time to cool it before it passed through. Thus a sudden shock, as of an explosion, may

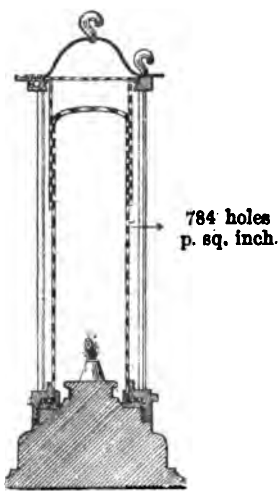


FIG. 350.—Davy lamp.

sometimes cause the flame to pass through the gauze. If the lamp is suddenly lifted into gas, there will be an explosion inside the lamp; for that reason the lamp should never be intentionally raised rapidly in a mine. The effect of the explosion is to force the burning gases inside the lamp against the gauze. The larger the lamp the greater the force of the explosion, and if the lamp were too large, the explosion would force the flame through the gauze; for that reason the diameter of the gauze must never be larger than $1\frac{1}{2}$ inch. If part of the gauze is taken away, and glass or solid metal substituted, then, in case of an explosion inside the lamp, the gases have a less area of opening through which to escape, and must therefore go more quickly, and there is a danger of the flame being forced through the gauze; so that the proportion between the contents of the lamp and the openings in the gauze through which the heated gases may pass must be observed. The Davy lamp is not perfect for several reasons. It will freely burn in an explosive mixture until it is red hot, and then it will ignite the external gases. The flame readily passes in a current exceeding 6 feet a second, and sometimes in a current of less speed, even 4 or 5 feet a second; and the light it gives is very poor, owing to the fact that $\frac{7}{8}$ of the cover is opaque wire, and less than $\frac{1}{5}$ opening. The wire pillars that unite the lamp top and bottom, together with the oil-pot and the top to which the suspending ring is attached, also diminish the light, so that the illumination produced by a Davy is between $\frac{1}{13}$ and $\frac{1}{30}$ of that given by a standard sperm candle.

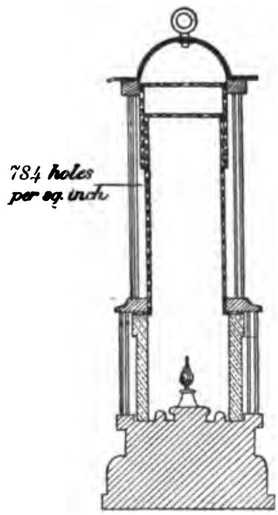


FIG. 351.—Clanny lamp.

Clanny Lamp.—The Clanny lamp (see Fig. 351) is similar to the Davy, but there is a glass cylinder instead of the lower portion of the gauze. The glass cylinder has a washer of soft leather, indiarubber, asbestos, or other flexible material at the top or bottom, so as to prevent a passage of air over it. A flange at the bottom of the gauze cap fits against a brass ring over the glass; the supply of air for the lamp enters through the gauze above the glass. The Clanny gives more light than the Davy, and can be more easily carried in an air-current without being blown out; it is superior in no other respect, whilst, if an explosion should take place inside, the burning gases, having a less area of gauze through which to escape in proportion to the contents of the lamp, would be more likely to fire the gas outside than in the Davy lamp. Such an explosion of gas inside a Clanny could, however, only take place if there was a very small oil-flame inside the lamp, permitting the lamp to be nearly filled with explosive mixture. If, on the other hand, the flame of the lamp was at its full height, a great part of the interior would be filled with the products of combustion from the lamp-flame, and therefore there would only be room for a small quantity of explosive gas. Therefore, if the Clanny lamp is used for testing for gas, it is in the first instance desirable to use it with the flame at full height; if this gives no indication of gas, then the flame may be reduced, and a more delicate test applied.

Stephenson Lamp.—The Stephenson lamp (see Fig. 352) differs from the Davy in the following respects: The gauze has a diameter of about 2 inches; inside is a glass which reaches

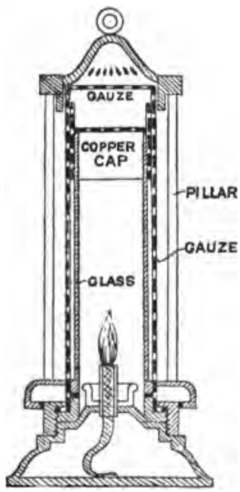


FIG. 352.—Stephenson lamp.

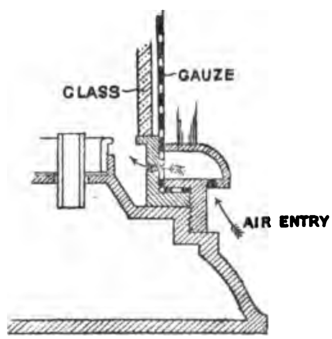


FIG. 353.—Stephenson lamp, enlargement.

nearly to the top of the gauze; the top of the glass cylinder is covered with a perforated copper cap, and it rests upon a brass ring. The air to feed the flame enters the lamp through a number of small holes about $\frac{1}{20}$ inch in diameter (see Fig. 353) in a hollow collar; it then passes through the wire gauze, which is fastened to the lower flange of this collar. The amount of air that can enter the lamp is thus regulated by the size of the inlet orifices. The lamp above the flame is mostly filled with the products of combustion; if, therefore, instead of fresh air, fire-damp should enter the lamp by these holes and ignite at the flame, the oxygen inside is instantly burnt up. For lack of this necessary element the flame goes out, so that, should such a lamp be left burning in gas, it would quickly extinguish itself. If, however, the lamp is exposed to a current of inflammable gas and air, the pressure forces a sufficient supply of oxygen inside, and the flame continues till the lamp is red-hot. In a current of from 8 to 12 feet per second the Stephenson lamp is unsafe.

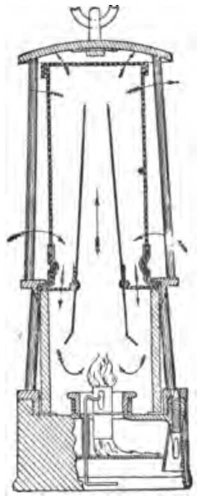


FIG. 354.—Mueseler lamp.

Mueseler Lamp.—This is a Belgian lamp (see Fig. 354), and may be described as a modification of the Clanny. There is a metal chimney by which the products of combustion are conducted to the upper part of the gauze cover. There is a horizontal diaphragm of gauze covering the lamp at the top of the glass, the only opening being where the chimney passes through, the gauze being tightly fastened to the chimney. The air enters the lamp through the outer gauze above the glass, descends through the horizontal gauze to the flame, ascends the internal chimney, and escapes through the upper part of the outer gauze. This lamp will burn brightly on account of the regulated air-current. It is also safer than the Clanny, because the entering air has to pass through two gauzes, and the escaping air has to pass up a chimney full of burnt air, which cannot support combustion, and then through the outer gauze; and also, if the lamp is held on one side, the flame, instead of cracking the glass, will go out. The lamp will go out if held in inflammable gas. The Mueseler lamp must always be carried by the ring at the top, and suspended in a vertical position. This lamp is unsafe in an explosive current of 15 feet a second.

Marsaut Lamp.—The Marsaut lamp, the invention of a well-known French mining engineer, only differs from a Clanny in having two gauzes (see Fig. 355) or three gauzes, each gauze tending to increase the safety of the lamp. On the other hand, the multiplication of gauzes interferes with the supply of fresh air to the flame. M. Marsaut was also the first, or one of the first, to make a shielded lamp.

Shielded Lamps.—Owing to the discovery that the Davy and other lamps were not safe in a rapid explosive current, it has been rendered obligatory by law to use a lamp which will resist a rapid current of explosive mixture without allowing the flame to pass. In the case of the Davy lamp, this has been achieved by placing it in a tin can (see Fig. 356). In the bottom of the can are perforations for the admission of air; in front is a piece of glass; the top is open, with a handle. A screw-lock keeps the lamp in its place. The Clanny, Mueseler, and Marsaut are shielded by an iron (tinned iron) covering, sometimes called a "shield," and sometimes a "bonnet" (see Figs. 357, 358). This protects the gauzes from the air-current.

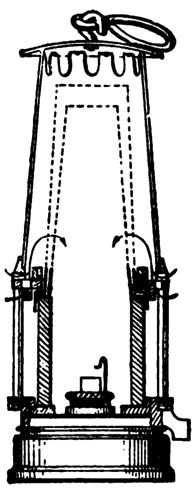


FIG. 355.—Marsaut lamp.

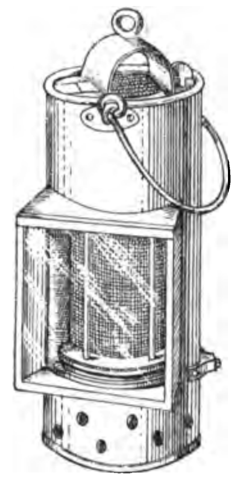


FIG. 356.—Tin-can Davy.

Well-made Mueseler or Marsaut lamps with this shield will resist a current of 30 feet a second. The shield is sometimes permanently fixed on the lamp. The objection to this is that the miner or deputy who examines the lamp in the pit cannot see if the gauze is in its place or not. To meet this difficulty the cover is sometimes made movable so that it can be removed at will in order to inspect the gauze. In other cases the cover is put on after inspection of the gauze, and is then locked. Fig. 359 shows a shield of corrugated metal, which has a spring lock. After inspection by the collier and deputy, the cover is placed over the lamp and cannot be withdrawn unless the lamp is unlocked and taken to pieces, and then, by means of a round wooden ram (Fig. 360), the spring is pushed back so that the cover can be removed. Many ingenious inventions have been made to increase the safety of lamps when exposed to rapid currents.

Evan-Thomas Lamp.—The Evan-Thomas is similar to a shielded Clanny, but the brass ring covering the glass is carried up some distance outside the gauze, and has a flange at the top. This lamp will resist a very great velocity; the only objection to it is that the flame is rather dull.

Morgan Lamp.—The Morgan lamp (see Figs. 361, 362) has a metal chimney to improve the draught, and the flame gives

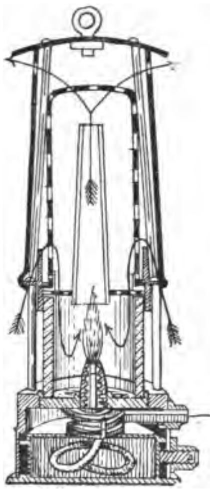


FIG. 357.—Section of Patent "Protector" Mueseler with shield.

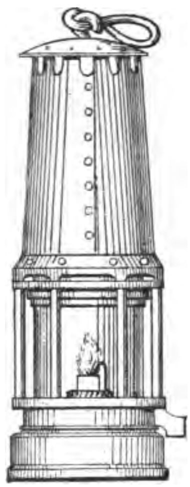


FIG. 358.—Shielded lamp (Marsaut): elevation.

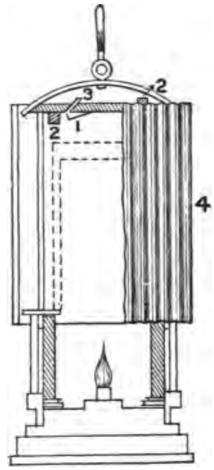


FIG. 359.—Marsaut with patent bonnet. 1, steel spring holding projection 2; 2, projections on corrugated bonnet, held in position by spring; 3, slot into which spring is pushed to allow bonnet to be taken off; 4, corrugated bonnet.

a good light. The inlet air enters through three gauzes, and the outlet air passes up the chimney and through two gauzes. There is a double iron shield, so that the air cannot blow straight through the orifice against the gauze, but is deflected in its passage. This lamp successfully resisted tests up to a velocity of 53 feet a second, and is sufficiently safe for every practical purpose.

FIG. 360.—Ram to open lamp.

Clifford Lamp.—The Clifford lamp (Fig. 362*a*) has a metal chimney terminating with a glass bell over the flame; it also has a double shield, the air-current being so deflected in passing through as to produce an equal pressure all round the lamp. The air passes through a perforated compound plate of copper and fusible metal, made so that if the lamp gets hot, the melting of the fusible metal should close up the air-holes. This lamp has successfully resisted every effort to explode it up to a velocity of more than 100 feet a second.

Protector Lamp (see Fig. 357).—This is a spirit-lamp burning a mineral spirit called colzalene, similar to benzoline. The oil-pot contains a sponge which absorbs the spirit, the surplus

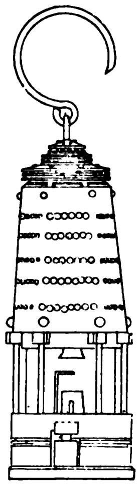


FIG. 361.—Morgan lamp.

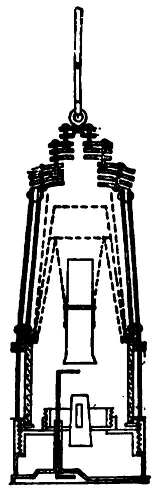


FIG. 362.—Morgan lamp: section.

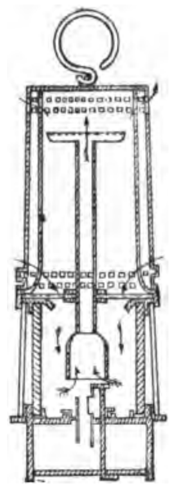


FIG. 362*a*.—Clifford lamp.

liquid being poured back into the reservoir. The wick of cotton-wool or asbestos does not burn. The protector construction of lamp is applied to the Davy, Clanny, Stephenson, Mueseler, and Marsaut lamps. The lamp is so constructed that it cannot be opened without being extinguished. To effect this, the wick-tube is surrounded with a brass tube, which is screwed into a brass plate which covers the bottom of the lamp below the glass. The oil-pot has a long wick-tube screwed inside the extinguisher tube. When the lamp has been lighted, a bolt F is pressed in; the fork at the end of it clasps the extinguisher tube and prevents it from turning; a spring on the bolt prevents its withdrawal until the lamp has been opened. The oil-pot can only be withdrawn by unscrewing. The wick-tube is in that way drawn down within the extinguisher tube, and the flame put out. The height of the flame can be regulated by unscrewing the oil-pot for a little way. Another lock prevents the lamp from being opened; this second lock is one of the usual kind, and is adopted as a second precaution.

Evan-Evans.—This is an automatic-extinguisher lamp; if the lamp should get hot, a solder joint is melted, and a spring forces down a metal cap which closes both the inlet and outlet for air in the bonnet, and so extinguishes the flame.

Wolff's.—This lamp (see Fig. 363) can be lighted without opening; it contains a reel of tape on which explosive composition is placed at intervals of about a quarter of an inch. There is a spring hammer A, operated by the button B, which strikes the explosive composition, the flash from which lights the lamp. The lamp is a spirit-lamp, otherwise it would not light by the flash. The height of the flame is regulated by the screw S, by which the wick-pipe is raised or lowered. This invention is ingenious, and the lamp is largely used in Germany, and is now used in this country. Attached to Wolff's lamp is a magnetic spring lock. By means of a spring a ratchet is pressed against teeth on the oil-pot, which prevents it from being unscrewed. The application of a powerful magnet, however, will withdraw the ratchet, and permit of the opening of the lamp.

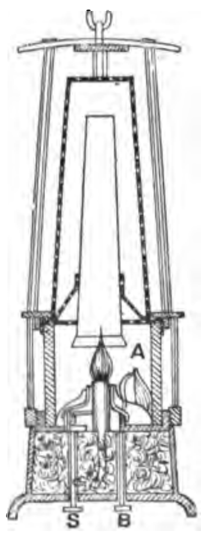


FIG. 363.—Wolff detonating lamp.

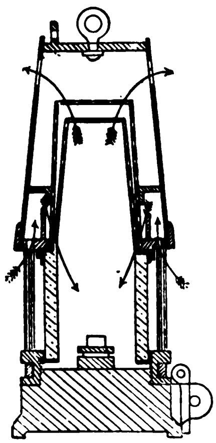


FIG 364.—Howat deflector.

Craig and Bidders.—This lamp was fastened with a spring lock, to be opened by means of a powerful magnet.

Deflector.—Amongst the numerous lamps recently designed to meet the requirements of the Mines Regulation Act, 1887, is the deflector (Fig. 364). In this lamp there are two gauzes, as in the Marsaut. The lower part of the gauze caps is surrounded with a brass ring, near the top of which are holes, through which air can enter the lamp. Above these holes a flange projects outwards towards the shield. The air-current enters through vertical holes in the brass ring covering the glass, as shown in the figure, and is then deflected downwards by the flange above named, through the gauze on to the flame. The products of combustion escape through the upper part of the gauze, and through openings near the top of the shield.

A. H. G. Lamp.—Another lamp that is used to a considerable extent by deputies, or fire-triers, is the Ashworth-Hepplewhite-Gray (Fig. 365). This lamp has a conical glass surrounding the flame, the air supplying which is introduced through a gauze below the glass, the products of combustion escaping through a gauze cap and metal chimney or shield above the glass. Four brass tubes extend from near the top of the lamp down to the brass ring below the glass, so that the air to feed the lamp is drawn from a stratum on a level with the lamp-top. So if this upper stratum contains gas, it will be detected by this lamp. There are, however, near the base of the tubes, openings closed by sliding shutters, so that the lamp can derive its supply of air from a lower stratum. The intention of the inventors is that the fire-triers shall first test the lower stratum of air, and, if that is clear, close the openings in the tubes, so as to suck air down from the top. They also recommend that the test be made without reducing the flame. The latest form of this lamp, as made by Mr. Ashworth, is shown in Fig. 365*a*. Here there is a shield, so that it can be carried in a rapid current of air.

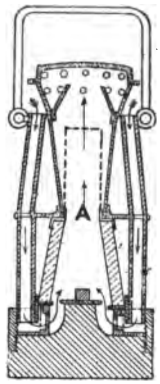


FIG. 365.—Ashworth-Hepplewhite-Gray deputy safety-lamp.

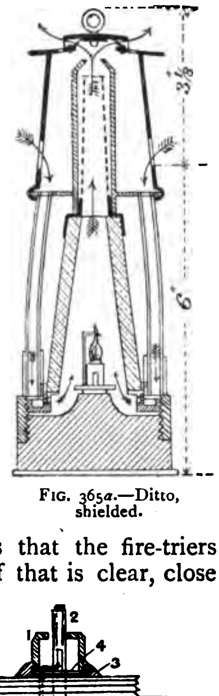


FIG. 365*a*.—Ditto, shielded.

Many safety-lamps are now made to burn paraffin instead of rape oil; the burner has to be modified for this purpose, as shown in Fig. 366, which is a drawing of the lamp used at the West Riding Colliery.

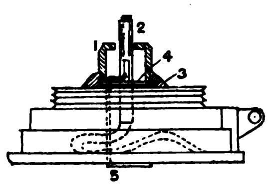


FIG. 366.—West Riding paraffin-lamp. 1, brass collar screwing down on to plate 4; 2, wick-tube with circular plate; 3, packing between circular plate and oil-well; 4, circular plate on wick-tube held down by collar; 5, pricker.

Thorneburry.—Owing to the small amount of light given by safety-lamps, a lamp has been made called the "Thorneburry lamp" (see Fig. 367),[1] which burns paraffin. It has a flat wick, giving a larger flame than the ordinary lamp. There is a double glass, and down the intermediate space the inlet air descends to a hollow ring, whence it passes through wire gauze on to the lamp-flame; a metal chimney secures a good draught, and there is a metal shield. The entering air has to pass through three gauzes, B, A, and the horizontal gauze round the base of the chimney, and the escaping products of combustion through a long chimney and the gauze cover A.

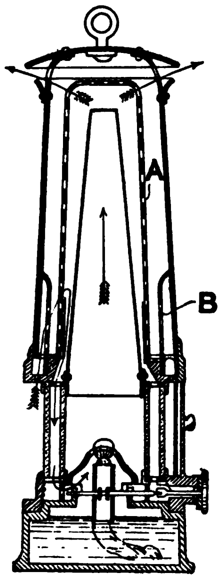


FIG. 367.—Thorneburry lamp.

It would require a volume of considerable dimensions to describe all the lamps that have been made, and many of great excellence are perforce omitted from this list.

Uses of the Safety-lamp.—One use of the safety-lamp is to permit workmen to enter a place containing explosive gases without getting burnt. Another use is to permit workmen to remain in a mine liable to sudden outbursts or blowers of gas without danger of explosion. A third use is to enable the fireman to ascertain that the mine is free from gas without risk to himself. Before the safety-lamp was made, the presence of gas was frequently ascertained by means of a candle, the flame of which perceptibly enlarges in gas. A very small percentage of fire-damp can be detected with a candle; but, unfortunately, the use of the candle is attended with great risk. When an ordinary safety-lamp with full flame is placed in air containing fire-damp, there is an elongation of the flame, and, if there is a considerable quantity, the lamp will be filled with the blue flame of burning gas. In testing for gas the lamp is generally raised, because the gas lodges in the higher parts of the mine, and in cavities which are not exposed to the air-current. For a minute examination, the flame of the lamp is drawn down until the yellow part of the flame is no bigger than a pin's head. If there is 2 per cent. of fire-damp in the atmosphere, a small blue cap may be observed. In drawing down the flame of an oil-lamp for such an observation, it is very likely to be extinguished. The spirit-lamp is more convenient for this purpose. By means of a screw, the flame can

[1] Instead of the burner shown in the figure, a vase-shaped porcelain burner (Barton's patent) is now generally used with this lamp.

be reduced till there is nothing but a small blue flame about $\frac{1}{8}$ inch high; 1 per cent. of fire-damp will be rendered visible in the shape of a very faint blue cap. Mr. Wm. Galloway, mining engineer, of Cardiff, made experiments as to the proportion of gas that could be discovered with an ordinary safety-lamp (see Fig. 368). The height of blue cap is marked in inches.

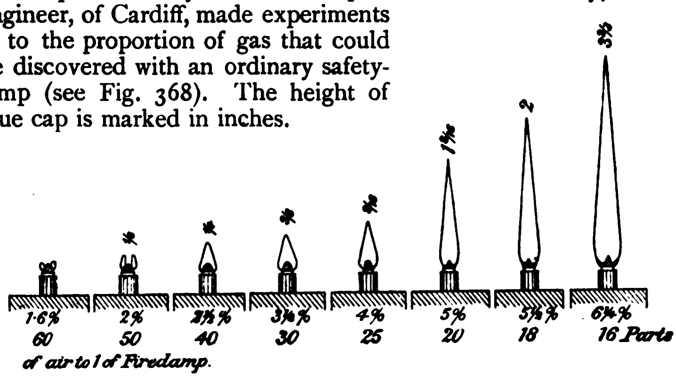


FIG. 368.—Blue cap on Davy lamp observed by W. Galloway in South Wales for eight mixtures of natural fire-damp and air. Lamp flame (reduced) shaded; blue oap unshaded; height of cap given in inches.

Pieler Lamp.—This is a German lamp (see Fig. 369), burning a non-luminous spirit, such as spirits of wine; there is a shield, *a*, which also protects the eye of the observer. When this lamp is put in a mixture of fire-damp and air, a very small percentage produces a blue cap; $\frac{1}{4}$ per cent. can be detected with this lamp. Owing to the fact, however, that the flame is so large, the caps given in a comparatively low percentage of gas are very long, and the lamp would rapidly become red-hot and unsafe.

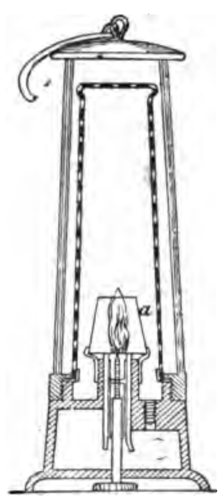


FIG. 369.—Pieler lamp.

Clowes Hydrogen Lamp.—This lamp, described in Chapter IX. (see Fig. 334), is specially constructed for the detection of small percentages of gas; $\frac{1}{5}$ per cent. can be clearly detected.

Lamp-room.—Whilst it is essential to have a lamp of a good pattern, it is no less important that it should be taken good care of. For this purpose, it is necessary to have a good lamp-room, and a sufficient staff, in the lamp-room, of experienced and responsible work-people, so that none of the parts may be improperly put together or omitted. With some of the shielded lamps now in use, it is quite easy for the lamp-cleaner to omit the gauze, which is essential to the safety of the lamps; and neither the workman who takes the lamp, nor the deputy who examines it, can very easily see the deficiency. This makes the examination by a deputy a mere farce, and it is obviously wrong that that should be the case. To avoid this, it is necessary, where such lamps are used, that the lamp should be given to the collier without the shield, which should be put on in his presence, and also in that of a deputy or an examiner especially appointed to see that the lamp is safe. This precaution involves more expense in giving out the lamps than has hitherto always been considered necessary, but it is a natural result of the use of shielded lamps. In order that too much time may not be consumed in giving out the lamps at a large colliery, it will be necessary to have a number of windows at which the workmen can take the lamps, and a number of lamp-men to distribute the lamps. Fig. 371 shows a design by the author for achieving this result.

Arc Electric Light.—This species of electric lamp gives far more light for a given expenditure of power than the incandescent

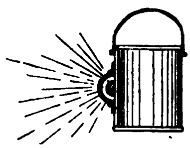


FIG. 370.—Swan portable electric lamp.

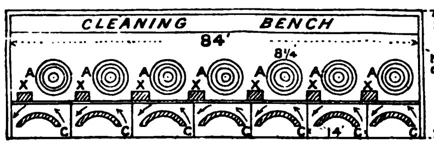


FIG. 371.—Plan of lamp-room. A, revolving tables on which lamps to be given out are placed; C, workmen's entrances; X, small tables on which lamps are put together and handed to the men.

lamp; it is, however, not a safety-lamp at all. It is largely used on the surface for lighting up railway sidings and other works; it is also sometimes used for lighting up large excavations in the pit. The author, in 1883, visited the Mechernich Lead-mine, where a large underground excavation, 200 feet long, 60 feet high, and 70 feet wide, was lighted by two electric arc-lamps, which gave a splendid illumination, and enabled the workmen to examine the sides and roof for loose pieces.

Electric Lamps.—Many portable electric lamps have been made; one by Swan (see Fig. 370). The electricity is supplied from a secondary battery, and the lamp will burn all day, giving a light of about half a candle; it weighs about 6 pounds. None of the numerous varieties of portable electric safety-lamps are yet in general use. There are, however, a great many fixed incandescent lamps used underground. These lamps are essentially safety-lamps, as the air is entirely excluded unless the lamp should be broken. If the lamp were broken in an explosive mixture it might possibly cause an explosion, but this is an improbable contingency. These lamps are now used to a considerable extent for lighting the pit porches, underground engine-houses, and, in some cases, engine-planes for long distances underground. The only practical danger in their use arises, not from the lamp, but from the conductors which convey the current; should these be too near each other, the current might pass between them and set fire to the timber; or should there be an explosive mixture, and the wire was severed, there would be a spark at the moment of severance, which would cause an explosion. These dangers may be reduced to a minimum by covering the conductors thickly with insulating material, and placing the positive and negative conductors at opposite sides of the road whenever practicable. Where the lamps are placed "in parallel," only a low voltage can be used, say 100 volts, and there is not much danger of the insulation being destroyed. If, however, the lamps are placed "in series," then a very high voltage may be employed, with a proportionate increase of risks.

Light.—The value of a safety-lamp depends to a great extent on the amount of light it will give, because a good light conduces to the safety, comfort, and earning power of the workmen. Dr. Court, of Staveley, has found that miners using safety-lamps suffer from a disease of the eyes called nystagmus, whereas those using candles do not suffer from it. The writer has made a number of observations to ascertain the amount of light given by various lamps. It must, however, be borne in mind that the light of a lamp varies greatly with the nature of the oil used, and the state of the wick, the cleanness of the glass and gauze, and the height of the flame. The illumination given by a candle is, from a collier's point of view, very satisfactory. There is seldom a strong current in the working place, so that the candle does not unduly sweal. The light given by a safety-lamp is only on a certain plane, a shade being thrown by the top and bottom, and pillars supporting the top; so that a lamp which, according to a photometric measurement made opposite the flame, gives a light equal to one-third of a candle, may in practical use only give the light of one-tenth of a candle, and it is probable that in many cases the light of a good candle is equal to thirty Davy lamps in ordinary condition with ordinary oil.

Swan's portable electric lamp gives a better all-round light than an oil-lamp, and is equal to about one-third of a candle on that side of the lamp where the light is fixed.

The following table shows the results of a number of photometric tests made by the author with safety-lamps :—

TABLE XII.

Light of safety-lamps compared with that of a candle.

Name of safety-lamp.	Number of lamps required to equal one standard sperm candle, the illuminated object being on a level with the flame.	Number of lamps required to equal one standard sperm candle, allowing for the shade cast by lamp-cover, bottom, and pillars. The candle and the lamp being each in a cylinder of white tracing-paper 2 ft. high and 8 in. diameter.
Davy lamp, very large flame ...	7 } 13 average	19 } 26 average
,, moderate ,, ...	18	27
,, ,, ,, ...	15	32
,, in tin can, good flame...	9	—
,, ,, flame rather low	25	—
Protector, shielded Marsaut, good flame	3	—
Protector, shielded Marsaut, flame average height	4¼	—
Clifford, very good flame	$1\frac{9}{10}$	—
,, moderate ,, ...	2½	5
Deflector, large flat ,, ...	3	—
,, moderate clear flame ...	5	—
,, small moderate ,, ...	6	—
Thorneburry, good flame	1¾	5
Ashworth, new pattern	3½	11½
Stephenson, good flame	7½	23
Tallow candle, good average flame...	1¾	—
,, large blaze	⅘	—

The following table gives the weights of various lamps without oil or wick :—

TABLE XIII.

Weight of safety-lamps without oil or wick.

Name of lamp.	lbs.	ozs.
Davy	1	10
Pieler	1	14
Stephenson	2	0
Tin-can Davy	2	4
Shielded Mueseler with reflector...	2	14½
Howat	2	15
Purdy's	3	0
Morgan	3	0
Clifford	3	4
Protector	3	4
Marsaut	3	4½
Clanny, with corrugated shield	3	5
Deflector	3	7
Thorneburry	4	2
Swan portable electric, charged (not made now) ...	6	0

CHAPTER XII.

BLASTING: WATER-CARTRIDGES, FLAMELESS EXPLOSIVES, LIME, WEDGING, ELECTRIC FUSE, ETC.

ONE of the most useful inventions of modern time is gunpowder; by its use a man can do work upon stone in an hour which without it would have taken him a whole day or several days. The use of gunpowder is very similar to the use of steam-power; in each case it is the combustion of some inflammable material which produces the power, and so relieves the workman of the greater portion of the mechanical labour, thus making his employment more intellectual. Gunpowder is made in a great many varieties, according to the work it has to do. For the purpose of the miner, it is often made in grains, about $\frac{1}{16}$ to $\frac{3}{32}$ inch in diameter. In this form the explosion is less rapid than in the case of smaller grains used for rifles and fowling-pieces; but, though less rapid, it is not less powerful in its ultimate effect. The reason for employing an explosive less rapid than rifle-powder is to give the rocks subjected to its action a little time in which to yield. When this time is accorded, the effect of the explosive extends to a greater distance from the blasting-hole than would be the case with a rapid explosive.

Fig. 372 shows a case in which a horizontal hole is drilled to a depth of 2 feet 6 inches, and charged with powder; the effect

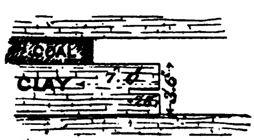


FIG. 372.—Horizontal shot at bottom of seam.

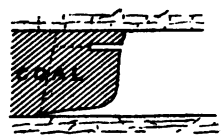


FIG. 373.—Horizontal shot at top of seam.

of this shot is carried forward from the end of the blast-hole to a further distance of 5 feet. Fig. 373 shows a hole drilled horizontally near the top of a seam of coal, to a depth of say 3 feet.

The coal is broken off further in than the back of the hole to a distance of 1 foot, and for a length of 6 feet on each side of the blast-hole. Had a more rapid explosive been used, it is probable that the amount of coal got down would have been much less, whilst that coal immediately underneath the blast-hole would have been shattered. Fig. 374 shows a way of blasting down the roof of a gate. In such a case, the effect of the shot will be to break down the roof for the whole width of the gate, and very likely for some distance beyond the end of the hole.

Gunpowder is now frequently used compressed in cylindrical lumps with a hole in the middle (see Fig. 375); the diameter of these cylinders being such as to go easily into the hole. These are much liked, as there is no risk of losing any powder, or trouble with cartridge-cases, as the compressed powder is itself a cartridge. This powder can be made to explode with the same rapidity as

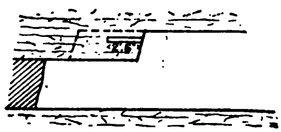


FIG. 374.—Shot in gate road.

FIG. 375.—Compressed powder.

ordinary blasting-powder. In hard ground more rapid explosives are necessary, because, in the case of ordinary blasting-powder, the gases produced by the explosion might escape through the fissures in the rock without breaking it; but in the case of dynamite, the explosion is too rapid to permit of the escape of the gases, and the rock is burst off. It is also possible to concentrate a much greater power of explosive upon a hard piece of rock by using dynamite; it is also preferred in wet ground. Dynamite is largely used in metalliferous mines, owing to the hard nature of the rock. One of the advantages of its use is the saving in labour, because only a small hole, say 1 inch in diameter, is required, as against a hole of 2 inches in diameter for gunpowder.

There are a variety of compounds, somewhat similar to dynamite, such as gelignite, blasting-gelatine; and other explosives, such as roburite, carbonite, ammonite, securite, bellite, tonite, gun-cotton, etc. Dynamite is a compound of nitro-glycerine and absorbent siliceous earth, called kieselguhr. Nitro-glycerine, which is an oil, was formerly employed by itself; but its use was dangerous, because the oil was liable to escape through crevices in the ground. Mixing it with kieselguhr makes a compound rather less powerful than the original, owing to the weight of useless material added. Blasting-gelatine is a compound of nitro-glycerine and gun-cotton, and is pernaps the most powerful explosive in ordinary use. Whilst the use of gunpowder is a great aid to the miner, its use is attended with some danger where there is fire-damp or coal-dust; and in most collieries both fire-damp and coal-dust are to be found in the working-places.

Hand-wedge.—To avoid this danger, many substitutes are employed. The most common substitute for gunpowder in coal-getting is an iron or steel wedge (see Fig. 376), which is used in the following manner. A hole is made 3 inches deep with the point of a pick, the point of the wedge is inserted, and it is driven in with a hammer; when the wedge is driven in, if the coal is not broken down, other wedges are inserted until the coal is broken off. But the application of a wedge in one part of a great mass of coal is apt to break it off in little pieces; in order to break it down in large pieces, a number of wedges should be arranged in a row (see Fig. 377), the number varying from six to twenty, according to the size of the pieces of coal broken off. Each miner may drive in two or three wedges, striking them alternately, so as to have an equal pressure on each. In this way the coal is broken down in large lumps. But, in order that a short wedge of this description may be effective in getting the coal in large pieces, it is necessary that the coal should be hard or tough; if, on the other hand, the coal is tender, the wedge will only break off small pieces near to it. To overcome this difficulty, an arrangement of wedges known as the "plug-and-feather" is used. An improvement called the "multiple wedge," patented by Mr. Elliot, is shown in Fig. 378. For the use of this wedge, a circular hole 2 inches in diameter has to be drilled with the drilling-machine; two tapered pieces of steel are put into the hole with the thick end first, between these two long wedges are driven, and then a third wedge between the other two. The effect of this is to bring pressure on to the back of the hole, and to break down the tender coal in large lumps.

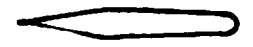


FIG. 376.—Wedge.

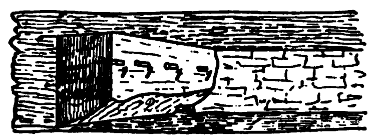


FIG. 377.—Breaking down coal by hand-wedges in a row. 1, wedges driven into coal; 2, undercutting.

Hydraulic Rams.—Hydraulic machines were introduced five and twenty years ago for breaking down coal, but, after being in use a few years they have been abandoned. One of these was designed by Mr. Chubb. He bored a hole 5 inches in diameter, placing in it a strong steel cylinder. This cylinder was perforated with a longitudinal hole, and at right angles to the axis of the cylinder were a series of rams about 2 inches in diameter; when

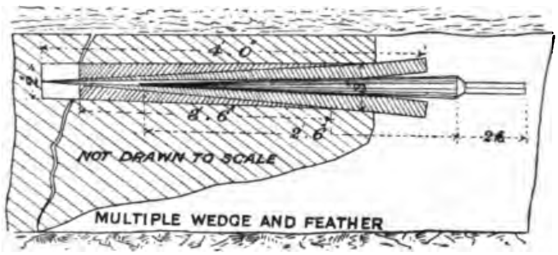
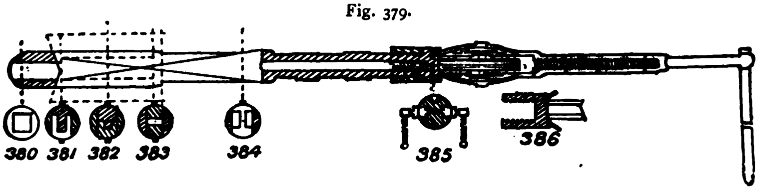


FIG. 378.—Elliot's multiple wedge and feather.

water was forced into the central cylinder the rams came out, and by their pressure broke down the coal.

Hydraulic Wedge.—Mr. Grafton Jones also patented several similar machines and a hydraulic wedge. This wedge, after a good deal of experimenting, was finally made in the form shown in Figs. 379–386. The drill-hole varied from $2\frac{3}{4}$ to $4\frac{1}{2}$ inches diameter, according to the size of the wedge. On reference

Fig. 379.



FIGS. 379–386.—Grafton Jones wedge.

to the figure, it will be seen that the wedge is forced between two inclined blocks, which move—one upwards, and the other downwards—equally throughout their whole length as the wedge is forced between them. The blocks cannot recede in the direction the wedge is moved, because tension bars hold a steel stop at the end of these blocks, so that they are forced to move up and down at right angles to the direction of the movement of the wedge. They and the wedge are of the hardest possible steel, polished and greased to reduce the friction. The wedge is forced in by a hydraulic ram, in a cylinder with an area of 2 square inches; that ram is driven forward by the water from a smaller ram, the area of which is 1 square inch. This smaller ram is forced along its cylinder by means of a screw; thus the power of the screw is multiplied by two, the difference in the size of the rams, and again by the slope of the wedge. How much power was absorbed in friction was never ascertained. Had there been no loss by friction, the pressure put upon these blocks would have reached 150 tons. The machine was applied to breaking down coal (see Fig. 387) that was exceedingly strong, and in some cases blocks of coal which had not been holed were forced out of the solid (see Fig. 388). Another hydraulic wedge was patented by Messrs. Bidder and Jones, and was for some time used at the Harecastle Mines in North Staffordshire.

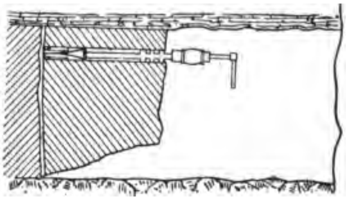


FIG. 387.—Wedging-machine.

These hydraulic machines were, some of them at any rate, capable of breaking down any coal; but they were not largely used, partly on account of the first cost, partly on account of the cost of the labour required in drilling the hole, and partly on account of the labour required in pumping, so that they could neither compete with gunpowder nor with the old-fashioned hand-wedging. If steam-power in some form had been applied to the drilling and pumping, they might have been more successful.

FIG. 388.—Grafton Jones wedge: application.

Hand-drilling Machines.—Various machines were used for drilling the holes. One of these, designed by the writer in 1869, is shown in Fig. 389. The drill is a piece of twisted steel, the twist clearing the hole of dust; it is fixed on to the end of a long screw, which passes through a nut fixed to an upright standard. A longitudinal key-bed was planed the length of the screw. Over the screw was put a small bevel wheel, a projecting

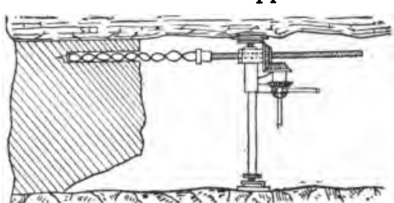


FIG. 389.—Drilling-machine.

key from which fitted into the bed on the screw, so that when the bevel wheel was turned the screw had to turn with it, and therefore it advanced through the nut. Two handles, arranged so that two workmen could work at once, gave movement to the bevel gearing; sometimes a small fly-wheel was used to help on the work. With this machine a hole could be bored by one man in soft coal on the "bord" with hardly any conscious effort, and with great speed; but when boring in hard coal the labour was severe. It was also hard to bore a hole on the "end." Since this machine has been made, many others have been introduced. One of these, known as the "villepigue perforator" (see Fig. 390), has been considerably used. It is not so suitable for the coal face, because the handle for turning the screw is at the end of the

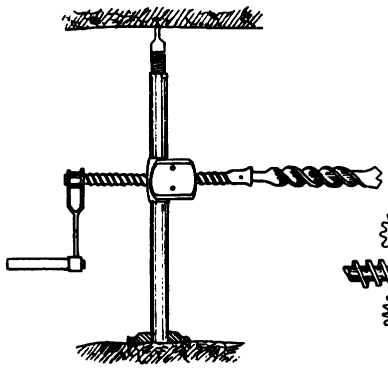


Fig. 390.

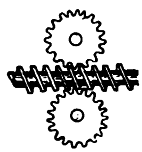


Fig. 391.

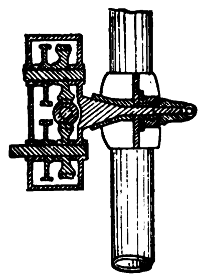


Fig. 392.

FIG. 390.—Villepigue perforator. FIG. 391.—Ditto nut. FIG. 392.—Ditto nut and breaks.

screw, and it frequently happens that there is no room at the coal face for such a machine to work. The nut in which the screw advances is made, as shown in Figs. 391 and 392, by the teeth of two of the wheels, which are kept from turning by a brake strap on a wheel (Fig. 392). If the drill encounters a hard lump, the wheels forming the nut will turn, so that the advance of the drill is stopped until the hard lump has been slowly scraped away, when the drill will again advance in the softer ground. This machine will readily bore holes in ordinary coal-measure shale, and will bore a hole 1 inch in diameter in exceedingly hard shale. Another variety of hand-drilling machines is shown in Fig. 393. In this machine the screw advances through a nut which is held firm by an iron bar, either curved or straight, one end of which is in a hole in the coal.

Lime Process.—Another plan of getting down the coal was patented about eleven or twelve years ago by Messrs. Smith and Moore; this is known as the lime process (see Fig. 394). When quicklime is mixed with water it rapidly heats and swells, and steam is given off; this expansion and evolution of steam was

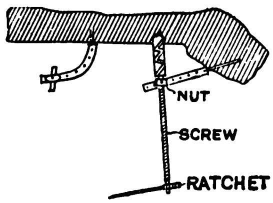


FIG. 393.—Hand-drilling machine.

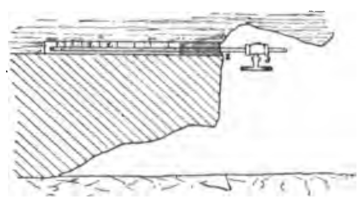


FIG. 394.—Lime process of breaking down coal. 1, lime; 2, tamping; 3, water-tap and pipe.

used to produce the pressure required to break down coal that had been holed or undercut. The hole was drilled about $2\frac{3}{4}$ inches in diameter, and into this were placed cartridges of compressed lime. The lime used was of the purest kind; it was ground to a fine powder, and then compressed by means of a hydraulic ram into cylinders about $2\frac{1}{2}$ inches in diameter and $4\frac{1}{2}$ inches long. Seven of these cartridges, occupying a length of 2 feet 8 inches, were put into one hole. Each cartridge had a groove moulded in it; a small metal tube was placed in the hole fitting into this groove. The lime cartridges were then stemmed with clay as if for a blast of gunpowder; at the outer end of the tube was a tap. About half a dozen holes would be charged simultaneously at a distance of five feet apart, and then by means of a force-pump a little water would be forced down each tube, and the tap instantly closed. In one or two minutes the water has taken effect upon the lime, and the pressure is brought upon the coal. This process was tried in mines in most districts of the country and abroad, but it was only found to be effective where the coal was easily broken down. Miners were not very favourable to its use, because quicklime is injurious to the skin, and, if the tamping should be blown out, a man might be injured by the escape of the lime and steam; and it was difficult, also, to keep the lime from getting mixed with the coal. These causes, together with the expense, have hindered the adoption of the process.

Air-shell.—Another ingenious process is the air-shell. For this a hole is drilled in the coal, and into it is pushed a cast-iron shell $11\frac{1}{4}$ inches long, $3\frac{1}{16}$ inches diameter outside, and $\frac{1}{2}$ inch thick. Attached to this shell is a strong tube which comes outside the hole, and is connected with a pump. The hole is closed by careful tamping. The pump is capable of compressing air up to a pressure of 10,000 lbs. on the square inch, or more, until the cast-iron shell was burst; the bursting of this shell breaks down the coal. In order to avoid the labour of compressing the air with the hand-pump, it was intended to compress the air on the surface with the steam-engine into very strong steel cases, which would then be carried down the mine and connected to the shell; by opening the taps the pressure would be suddenly brought into the shell to burst it. In this case the cost of the cast-iron shell, apart from the other expenses of the process, would probably be enough to prevent its adoption.

Water-cartridges.—In the year 1882, October 9, Mr. McNab read a paper at a meeting of the North Stafford Institute of Mining Engineers on "A Means of extinguishing the Flame of a Gunpowder Blast," so that it might be safely used in a coal-mine. Mr. McNab's method was to place the cartridge of gunpowder inside a case full of water, and he suggested that the water extinguished the flame. Further experiment however, showed that this was not always the case. Attention being called to the subject, it was discovered that with some other explosives the water did extinguish the flame. Cartridges of dynamite or tonite, etc., were placed in the centre of a waterproof bag (see Fig. 395) filled with water, and upon explosion the force of the cartridge was found to be at least as effective as if there had been no water, and it was thought by some people that the water increased the useful effect of the explosion, whilst at the same time no flame could be observed. It is considered that a high explosive used in a water-cartridge is safe. It appeared, however, to be possible, if not probable, that the water might escape from the bag before the discharge of the shot, which would then be dangerous. To overcome this difficulty, Messrs. Heath and Frost invented the use of a soapy jelly which could not run out of the bag, and which was quite as effective as pure water in stopping the flame. Many other means of stopping the flame have been used; one described by Mr. William Hargreaves in a paper read before the Midland Institute. In this case, a material (Trench's compound) composed of sawdust soaked in some chemical (aqueous) fluid was put round a cartridge of tonite, with the object of killing the flame.

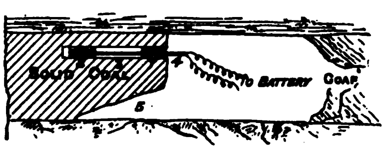


FIG. 395.—Water-cased cartridge. 1, tamping; 2, water surrounding cartridge; 3, dynamite cartridge and detonator; 4, electric wires for firing; 5, holing.

Flameless Explosives.—Other inventors turned their attention towards the discovery of a *flameless* explosive, so that water-cartridges and other *flame-killing* devices should be unnecessary.

Several of these have been very largely adopted, and are now manufactured on a large scale. A great number of experiments have been made, showing that they could be discharged into an explosive mixture of fire-damp or coal-dust, or both combined, without inflaming them, and that when fired in a pit no flame could be observed. By others it is, however, alleged that flame has been observed. However this may be, there is no doubt that the amount of flame is very much less than in the case of the admittedly flame-producing explosives, and it is considered safe to use these explosives where gunpowder would be dangerous. Among the flameless explosives which have been made are: ammonite, amvis, bellite, dahmenite, electronite, Faversham powder, pembrite, roburite, westfalite. In addition to the above, there are various "permitted" explosives made with nitroglycerine, and others composed chiefly of the ingredients of ordinary gunpowder, which, on being tested, give much less flame than the old-fashioned blasting powder. The balance of the evidence obtainable is in favour of the explosives made with nitrate of ammonium, in regard to safety in gassy and dusty coal-mines. The Secretary of State from time to time publishes a list of permitted explosives. The orders dated July 24, 1899, are given in the Appendix in full, and show the composition, and rules regulating the use, of all the "permitted explosives." Some of these explosives require careful handling, because some of the constituents are poisonous and may be absorbed through the skin, and, as in the case of ordinary blasting powder, the fumes are injurious, and the workmen should not return to the place after a shot until the ventilation has had time to clear them away.

The rules to be observed in blasting, in order to keep within the provisions of the Mines Regulation Act, 1887, must be taken from the general rules, but the following is a summary of some of the principal provisions:[1]—

The scraper and rammer must be of brass or copper, or must at least have a brass or copper end.

The pricker (if any) must also be of brass or copper.

No coal-dust must be used in ramming the shot, but such materials as clay and shale.

The explosive must not be forcibly pressed into a hole of insufficient size; and when the hole has been charged, the explosive must not be unrammed; and no hole should be bored for a charge at a distance of less than 6 inches from any hole where the charge has missed fire.

An explosive should not be stored in the mine, and it must not be taken into the mine except in cartridges in a secure case or

[1] See Appendix, Explosives in Coal Mines Orders.

canister, and containing not more than 5 lbs., and a workman must not have more than one of such cases or canisters in one place at the same time.

In any place where the use of locked safety-lamps is for the time being required, or which is dry and dusty, no shot should be fired except by a competent person specially appointed for the purpose, who must examine the place and all contiguous accessible places within a radius of 20 yards before firing the shot.

In cases where, at either of the four inspections last recorded, inflammable gas has been found in the same ventilating district, then the shot must not be fired unless the place where the gas was found has been cleared of gas, and there is not sufficient to be a source of danger; or unless the explosive employed in firing the shot is so used with water, or other contrivance, as to prevent it from inflaming gas, or is of such a nature that it cannot inflame gas.

If the place where the shot is to be fired is dry and dusty, then the shot should not be fired, unless the place of firing and all contiguous accessible places within a radius of 20 yards from it are, at the time of firing, in a wet state, or have had a thorough watering in all parts where dust is lodging, whether roof, floor, or sides; or, in places where watering would injure the roof or floor, unless the explosive is so used with water or other contrivance as to prevent it from inflaming the gas or dust, or is of such a nature that it cannot inflame gas or dust; but if the dry and dusty place is part of a main haulage road, or is contiguous thereto, and showing dust adhering to the roof and sides, unless the place shall have been not only watered, but a flameless explosive used; or where either the water has been used, or the water-cartridge and flameless explosive only, and all the workmen have been removed from the seam in which the shot is to be fired, and from all seams communicating with the shaft on the same level except the men engaged in firing the shot, and such other persons, not exceeding ten, as are necessarily employed in attending to the ventilating furnaces, engines, machinery, winding apparatus, signals, or horses, or in inspecting the mine.

CHAPTER XIII.

POWER DRILLS AND HEADING MACHINERY.

ALL underground works, whether sinking, driving levels, tunnels, or excavating masses of rock or ore, where the rock or mineral is very hard, require the use of power drills for economical work. The necessity for such drills was forced upon the engineers of the Mont Cenis tunnel five and thirty years ago. Before these were adopted, the progress of the tunnel was very slow; with the use of these machines, the speed of from 8 feet to 13 feet in twenty-four hours was attained. For this work Sommeiller's drill was used; this was a large and heavy machine, not often seen nowadays. Since that time a great number of excellent machines have been made, of which one or two specimens are here described; but at the same time, it must not be supposed that the prominence given to these machines indicates that they are any better than the others not mentioned. One of these, called the Climax drill (see Figs. 172, 173), has a cylinder say from 3 inches to 4 inches in diameter, stroke of the piston about 4 inches; a tappet on the piston-rod gives movement to an oscillating slide-valve; there is no cushion of air to prevent the drill from striking with full force. Fig. 396

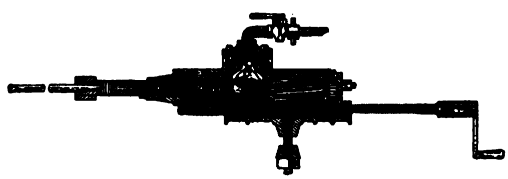


FIG. 396.—Rio Tinto power drill.

shows the Rio Tinto drill; a tappet on the piston-rod gives movement to a piston slide-valve by means of an oscillating forked lever. The Darlington drill (see Fig. 397) has no valves; the piston is very heavy, and as it moves, opens and covers the ports. The Minera drill also has no valves, and has a double piston. These valveless drills work very quickly and safely; the piston is cushioned at each end of the stroke, and therefore there is a tendency to strike a gentler blow than in the case of drills which have valves and not so much cushioning for the piston. In all these machines the feed is by hand; that is to say, the miner turns the screw which causes the machine to advance along the slide, which is carried

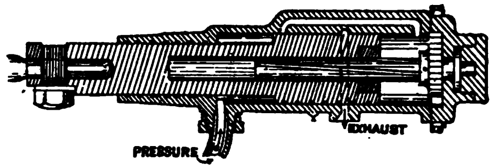
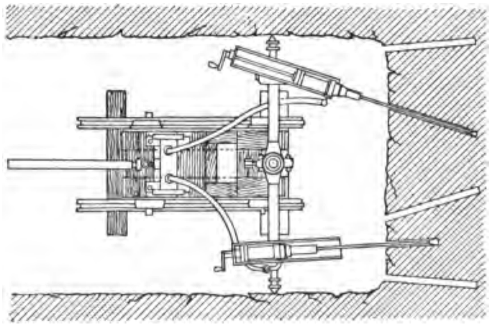


FIG. 397.—Darlington drill: section of cylinder.

on a tripod stand or fixed bar. The number of blows struck is say from 400 to 600 a minute; the dirt is cleared from the holes by a jet of water and the movement of the drill-rod; holes may be bored to a great depth by changing the drill-bar. The machines above named are not intended to bore holes much more than 6 feet in depth, and from 1 inch to 2½ inches in diameter. When the machine is fixed, the boring may be done at a speed of from 2 inches a minute up to 9 or 10 inches a minute in granite. To bore a hole 3 feet deep in fifteen minutes, including changing the drill-bar, would be good work for ordinary practice; it would very likely take two hours for three men to bore the same hole by hand.

The drills are driven by compressed air at a pressure varying from 30 to 60 lbs. per square inch. As a rule, the machine is placed on a tripod stand as shown in Fig. 172, and where the rock is hard the stand does not move; if, on the other hand, the ground is soft, the feet of the stand may move, and then the drill is apt to get jammed in the hole. For tunnel work the machines are sometimes fixed on a cross-bar; a similar arrangement is sometimes used for sinking. For rapid work in driving levels, the machines are fixed on an iron carriage (see Fig. 398). Two or four machines may be carried on this carriage, according to the size of the tunnel. When the requisite number of holes have been drilled, the machine is rolled back along the rails to a safe distance until after the blasts have been discharged and the dirt cleared away. The compressed air that drives these machines is useful for clearing out the smoke, and, if the mine is hot, for cooling the heading. Where every arrangement is made for speed, from 30 to 50 yards of heading may be driven in one week, where by hand it would have been difficult, with every exertion, to drive 6 yards a week. In ordinary mining work, however, great speed is not usually obtained, because of the capital required to provide the requisite machinery and the extra expense in labour, and a speed of from two to three times that of hand-labour is usually considered sufficient.



PLAN

FIG. 398.—Drills on carriage.

Rotary Drills.—A carriage carrying two rock-drills is shown in Fig. 399.[1] These drills are rotary, as in hand-boring machines, with a twisted steel auger; revolving motion may be given from

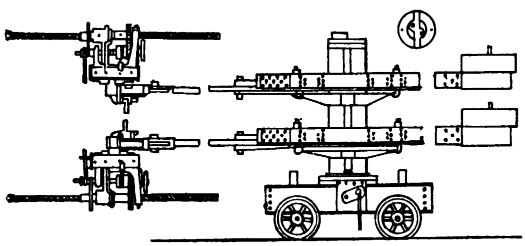


FIG. 399.—Walker's drilling-machine.

any source of power, such as a compressed air-motor, hydraulic motor, or electrical motor. A novel way of driving these drills is by the oil-engine, which is placed on the carriage, and a belt from the fly-wheel gives the required rotary movement to the drill. The writer has seen one of these machines drill a hole in Cleveland ironstone at the rate of one yard in one minute. It was driven by water-pressure.

Tunneling Machines. — Machines have been made for driving levels without the aid of explosives; one of these, called Brunton's, was set to work in a slate-quarry in North Wales. The power was conveyed to the machine by a wire rope, and this gave movement through gearing to a revolving head of iron; on it were

[1] The long horizontal arms carrying the drills and balance-weights are shown broken to reduce the length of the drawing.

placed steel discs with a sharp edge on the periphery. These discs cut away the rock, and a circular tunnel was made 4 feet 6 inches in diameter in a hard slate rock. As there was no blast to shake the ground and leave loose projecting pieces, a tunnel made by this machine in such ground requires no lining to make it safe. Another machine, invented by Colonel Beaumont, shown in Fig. 400, was used for the exploring heading for the Channel tunnel, which was driven in the chalk under the sea near Dover. A revolving head carried steel teeth which scraped away the chalk, the fragments of which were carried to the rear of the machine, and thus cleared away. The chalk is a very suitable material for such a machine, being so soft that it can be cut with a penknife, whilst, on the other hand, it is sufficiently tough to stand without lining. The diameter of the heading was 7 feet, and the rate of advance when in actual work $\frac{5}{8}$ inch per minute, which is equal to 3 feet an hour; this rate, of course, could not be maintained for long, having regard to the numerous stoppages necessary for the adjustment of the machine. The above machine was driven by compressed air. Another and similar machine, but much stronger, was used by Colonel Beaumont in tunneling under the Mersey, between Birkenhead and Liverpool, cutting a circular heading 7 feet 4 inches in diameter, and advancing on an average 17 yards a week, with a maximum of 34 yards a week; but, afterwards improved, it attained a speed of 54 to 65 yards a week, whilst the speed of driving by hand a heading 10 feet by 8 feet in the same rocks was from 10 to 13 yards a week.[1] The rock in this case was a sandstone of the New Red formation. Colonel Beaumont also used a similar machine at one or two other tunnels in different rock. A somewhat similar machine has been patented by Messrs. Stanley (see Fig. 401) for driving in coal; the machines are made suitable for driving tunnels either 5 feet, 6 feet, or 7 feet diameter as required. These machines have a revolving head, on to which are fixed projecting teeth or cutters upwards of 2 feet long, which cut a circular groove about 3 inches wide, leaving a solid block of coal in the centre of the heading. As the boring-head revolves, it is automatically advanced from the frame by a screw. After boring the depth of the teeth, the centre block is broken off and cleared away; another cut is then commenced, and so on as far as the traverse of the central screw permits. The frame, which is carried on two centre wheels, is then moved forward by reversing the movement of the screw, and cutting recommenced. The centre block often bursts off when only 9 to 10 inches has been cut. This machine will work at the rate of 3 feet an hour; the average speed

[1] The machine was inspected by the author whilst at work, but the particulars of speed attained are taken from a paper read at the Inst. Civil Engineers, May 4, 1886, on the "Mersey Tunnel," by Mr. Francis Fox, M.I.C.E.

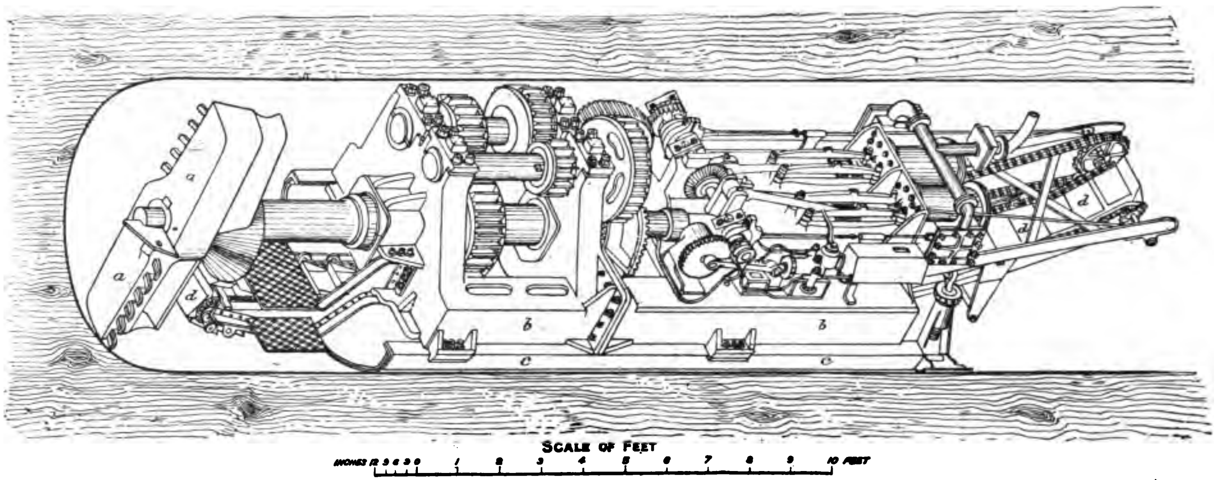


FIG. 400.—Section of the present heading under the sea at Dover, showing the air-boring machine of Colonels Beaumont and English in isometrical perspective.

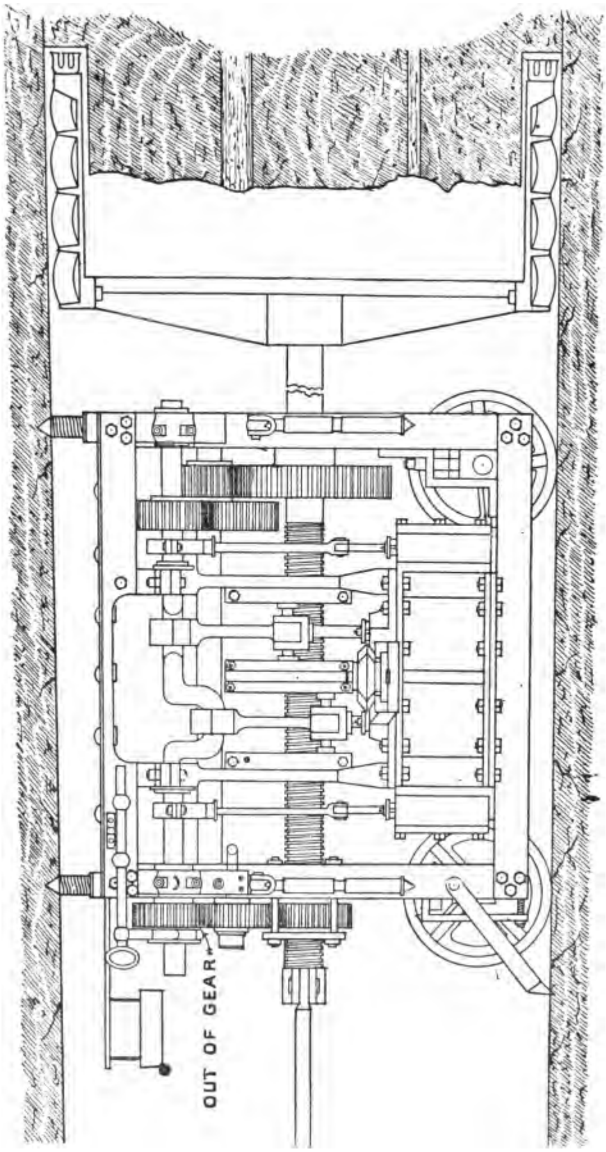


FIG. 401.—Stanley's coal-heading machine : elevation.

of heading attained is usually much less, but that is simply a matter of cost and organization. A speed of from 30 to 50 yards a week has been obtained in practice. The engines are driven by compressed air. By the use of these machines the shattering effects produced by gunpowder are avoided, and in some cases the headings will stand secure. The size of headings driven, however, is in many cases too small, and two machines have to be employed side by side (see Fig. 402), the cuts from which leave a little centre rib at top and bottom, which has to be got down. A two-headed machine is also made by Stanley, the circles made by the two revolving heads intersecting each other, so that no central pillar of coal is left between the two cuts. The head may be also widened in the manner described in the chapter on "Opening Out." Owing to the nature of beds of coal, fire-clay, and shale, the headings cut by these machines do not always stand without timbering, for which extra height may have to be got down. Where it is necessary to advance a pair of headings with great speed, and the ground is not too hard, there is no doubt that this machine will do good service; on the other hand, it must be borne in mind that if a heading is driven in fiery coal, at the rate of 20 yards a day, the amount of gas encountered may be unusually large, and require careful ventilation. The compressed air used for driving the machines will suffice to clear the heading, but the manager must remember that when this compressed air is shut off, from any cause, the heading may be rapidly filled with fire-damp.[1]

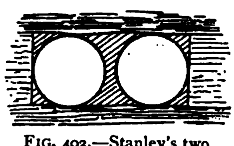


FIG. 402.—Stanley's two tunnels.

Messrs. Stanley have recently introduced a new form of their heading-machine, in which the coal or stone is all cut up into little pieces, so that the fragments can be gathered up, and carried behind by means of machinery, and thus the advance of the machine may be continuous; and they claim, for this reason, for the new machine twice the speed usually averaged with the annular groove machine.

Electric Drills.—Some electric power drills have been recently made, and were exhibited at the Frankfort Exhibition of 1891; one of these was the application of an ordinary motor to gearing, the movement of a crank being used to lift the drill, the blow being struck by the force of a spring. Another used the solenoid, by which a reciprocating movement is given to a bar of iron. As these have not yet come into general use, it will be unnecessary to give a further description.

[1] As it is undesirable ever to have an accumulation of fire-damp, it is necessary to provide for the *constant* ventilation of the heading independently of the cutting-machine.

CHAPTER XIV.

COAL-CUTTING MACHINERY.

PERHAPS the most arduous part of the work of a miner is holing or undercutting, called "kirving" in the north of England. In order to get the coal, it is frequently necessary to cut away either some of the lower part of the seam or some of the under clay for a distance of from 1 foot up to, in some cases, 8 feet, under the coal. The height of the excavation so made varies from 12 inches up to 2 feet 6 inches at the face, tapering down to a couple of inches at the back (see Fig. 403). This undercutting is sometimes done only for a short distance, say 2 yards, in the case of stalls in pillar-and-stall mines. One side of the stall is then nicked, and the other side of the piece of coal that has been holed is broken down by a shot. Sometimes the whole width of the stall is holed at once.

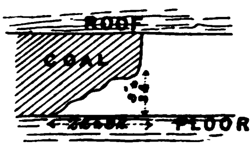


FIG. 403.—Section of holing.

In longwall stalls it is customary to hole a considerable length, say from 10 yards up to as much as 100 yards. Usually the holing only extends the length of one-half the stall, say 20 or 30 yards. During the process of holing, the coal is supported by sprags fixed at intervals of not less than 6 feet. When the sprags are withdrawn, the coal often tumbles down. There are generally joints in the coal parallel to the cleavage, and others at right angles to the cleavage, and if the holing is made as far in as one of these joints, there may be nothing to prevent the coal from falling when the sprags are withdrawn. In other cases the coal sticks to the roof, and has to be either cut, wedged, or blown down. The holing is sometimes done in dirt above the seam of coal, and sometimes it is done in the middle of the seam, especially where the seam of coal is divided by a band of dirt.

It is a long time since inventors first began to attempt to make machines to do some of the work usually done by the collier in the mine. Out of the great number of coal-cutting machines the following are some of the most notable.

Firth's.—This machine (see Figs. 404–406[1]) is worked by compressed air. An iron carriage on four wheels has an engine

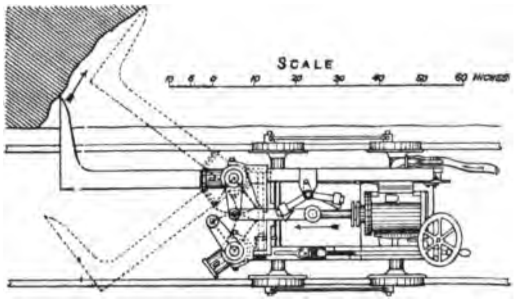
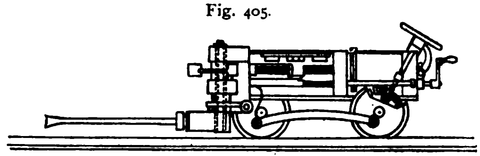


FIG. 404.—Plan of Firth's pick machine.

cylinder 6 or 7 inches in diameter; the piston-stroke is 12 inches long, and by means of a short lever turns a vertical shaft through an angle of 120°. Fixed upon the shaft is a steel pick, the point of which is about 3 feet distant from the centre of the shaft. As the engine moves, this pick strikes against the seam of coal. It can be so adjusted vertically as to work in the right place. The

Fig. 405.



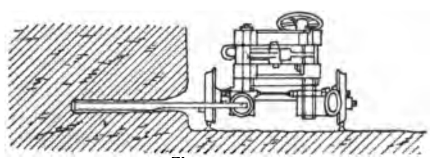


Fig. 406.

FIG. 405.—Firth's pick machine : side elevation. FIG. 406.—Ditto : end elevation.

valve is worked automatically by tappets on the piston-rod, but if the engine stops, or if the pick does not cut a full stroke, the valve can be worked by hand. The pick, which weighs 75 lbs., strikes 74 blows a minute. The point of the pick is a small piece of hardened steel fixed into the end of the pick; when it is blunt, it

[1] Proceedings Mechanical Engineers : Mr. Fernie's paper.

can be taken out and another one substituted in a minute or two. The wheels of the carriage are connected by a side rod, and by means of a hand-wheel and bevel gearing the carriage-wheels can be turned and the machine advanced.

The miner causes the machine to advance as fast as the machine can cut a slot in the coal. The height of the slot is about 3 inches, and this pick will undercut for a depth of 2 feet. The holing done has to be made deeper than this, so that, after the face of work has been holed to a depth of 2 feet, a longer pick, 4 feet 9 inches from the centre of the vertical shaft to the point, and weighing 90 lbs., is put on. This pick strikes in the slot already cut at the rate of 60 blows a minute, and cuts an extra 1 foot 9 inches under, making altogether 3 feet 9 inches under.

If it is necessary to make a deeper holing, a third pick, 6 feet long from the centre of the shaft to the point, is attached. By means of this the holing can be deepened another 15 inches, making the holing 5 feet under. Where the holing is not very hard the machine works at a great pace; in twenty-four hours, continuous working, 257 yards have been holed 3 feet 6 inches under. The machine will also cut in very hard ground, but at a reduced rate.

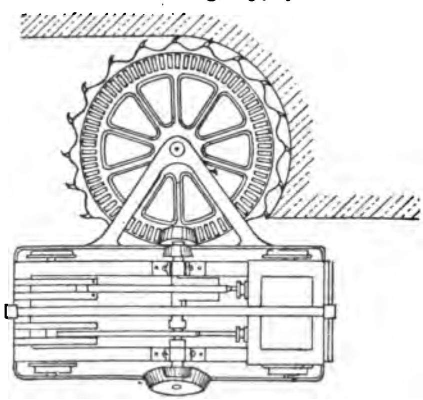


FIG. 407.—Gillott and Copley's coal-cutter.

Gillott and Copley's.—This machine (see Fig. 407) is similar to the last, in so far that it is driven by compressed air, and is carried on a carriage with four wheels running on rails parallel to the face of work. It has, however, a pair of cylinders each 8½ inches in diameter, with a stroke of 9 inches. These cylinders, by means of gearing, give movement to a revolving disc or wheel, on the periphery of which are fixed 20 steel cutters or picks. The speed of the engine is reduced by 5 to 1. This machine cuts a groove 3 inches deep. At the start, the groove has to be cut by hand for the wheel to enter. The pressure of the air used is about 40 lbs. It holes between 2 and 3 feet under, and will do 144 yards in sixteen hours of moderately hard holing.

Winstanley's.—Winstanley's machine (see Fig. 408) differs from Gillott and Copley's in this respect, that the cutting-wheel is carried on a movable arm, by which it can be held against the coal so as to cut its way into the seam, and so save having to make an entrance by hand-labour. It is driven by a pinion with very large teeth, which gear into large teeth on the periphery of the cutting-wheel; into these teeth are fixed the steel cutters. By this arrangement, as the lever carrying the cutting-wheel is moved it revolves round the pinion, which is always in gear. Diameter of cutting-wheel over cutters, 3 feet 9 inches; it holes about

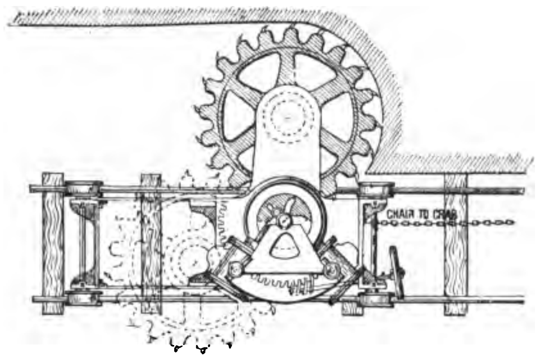


FIG. 408.—Winstanley's coal-cutter.

2 feet 3 inches under; two oscillating cylinders 9 inches in diameter and 6-inch stroke make 150 revolutions a minute. When all the cutters are sharp it will cut 1 square yard in a minute, with air at a pressure of 25 lbs.; but the average speed in moderately hard coal is about 8 square yards an hour. The space occupied by the machine is 3 feet wide by 7 feet long and 1 foot 10 inches high. The cutting-wheel, however, requires a width of 3 feet 8 inches in which to move without the cutters. This machine has done good work, but the writer is not aware that any are in use at the present time.

Rigg and Meiklejohn's.—Rigg and Meiklejohn's (see Fig. 409) is similar to Gillott and Copley's. The total height of the machine is only 16 inches, so that it will work in a very thin bed. There are a pair of cylinders 8 inches in diameter and 12-inch stroke; the cutter-wheel, 4 feet in diameter, has twelve cutters. It cuts 3 feet 6 inches under, and is self-propelling. When the cutters are sharp, it will cut 1 square yard in $1\frac{1}{2}$ minute; where the coal is hard, it will hole 5 square yards an hour on an average.

Baird's.—Baird's machine (see Fig. 410), instead of a revolving wheel, has an endless chain passing round a jib, and cutters attached to the links of the chain have been used.

Bower and Blackburn's.—This machine has a revolving

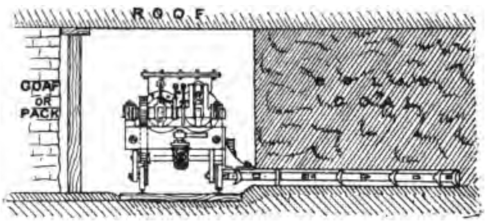


FIG. 409.—Rigg and Meiklejohn's coal-cutter.

arm or bar (see Fig. 411, elevation, and Fig. 412, plan) instead of a revolving wheel. 180 cutters are fixed on to this bar, which revolves at the rate of 500 revolutions a minute, holing 3 feet 6 inches under, the groove being about 4¾ inches deep. In easy ground it holes at a great rate, as much as 16 to 18 yards an hour. Like all other machines, it works much more slowly in hard ground. The motive power in this case is an endless rope passing round pulleys, as shown in the plan; this is ingenious.

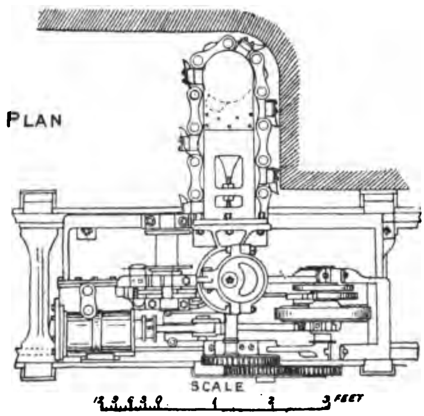


FIG. 410.—Baird's coal-cutting machine.

Electric Coal-cutters.—A modification of this machine is to substitute electricity for the rope. The armature of an electric motor is fixed on the cutter-bar, causing it to revolve at a great speed.

The writer has seen all the above machines at work, as well as others, many times during the last twenty-five years, having made a special study of coal-cutting machines.

Harrison's.—The Harrison coal-cutter (see Figs. 413 and 414) is an American machine, and is like a rock-drill. It is made in two sizes; the heavier machine, 700 lbs. weight, will undercut 4½ feet. The cutter attached to the piston-rod strikes repeated blows against the coal, making its way in. The machine is carried on two wheels, and when working is placed on some boards so that it can be easily turned. A workman sitting behind it can direct the cutter to any portion of the seam he desires. By moving the drill, he cuts away all the coal within reach, which is a segment of a circle. He then moves the machine on the boards, and subsequently moves the boards also, so as to carry the cutting along the face. The machine is driven by compressed air, and takes two men to work it, one man guiding it, and the other removing the fragments broken off by the cutter.

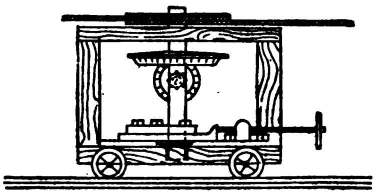


FIG. 411.—Bower and Blackburn's coal-cutter: elevation.

Yoch.—The Yoch machine is similar to the Harrison; it is driven by an air-pressure of 50 lbs.; the cylinder 6-inch diameter, 12-inch stroke. It is said that it will undercut from 60 to 70 feet in length to a depth of 4 or 5 feet in a shift of nine to ten hours. It weighs 1200 lbs. Two men are required to work it.

The advantage of these small machines is their portability and the ease with which the cutter can be directed to that portion of the seam where the holing ought to be made.

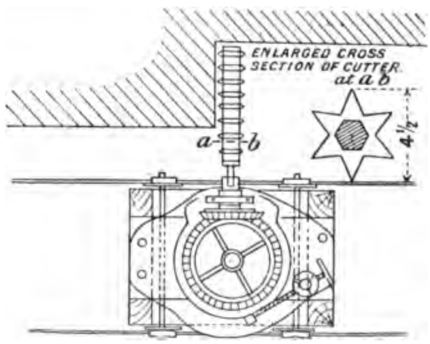


FIG. 412.—Bower and Blackburn's coal-cutter: plan.

Hydraulic Machines.—An ingenious machine was made about five and twenty years ago by Carrett and Marshall, and was thoroughly tested. It worked by hydraulic pressure, and had a slotting action. It was held in its place by a hydraulic jack, which jammed it tight between the roof and floor at each stroke; but the practical difficulties in the way of its use caused its abandonment.

Non-use of Machines.—It is thirty years since the pick machine was first introduced. Gillott and Copley's, Winstanley's, Baird's, and others, were brought out more than twenty years ago, but comparatively little undercutting is done by machinery. The difficulties in the way are various. Working places are often so well filled with packs of stone and with props that there is no room for a machine to pass along the face, whilst there is plenty of room for the miner. When the coal has been got down in a longwall face, rails are laid along it for the waggon into which the coal is filled; but as soon as all the coal has been sent out, the rails in ordinary course are pulled up, and their place occupied

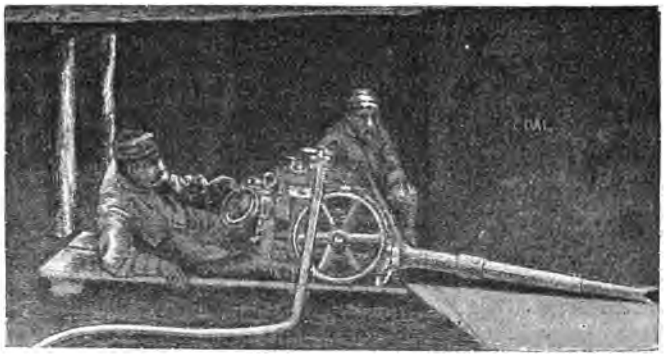


FIG. 413.—The Harrison coal-cutter at work.

with packs and props. But if a coal-cutting machine is to be used, then the packs and props must remain at least 4 or 5 feet back from the face, and in some seams the roof may break down. Whatever precautions are taken, the roof is liable to break down, and if a machine is there it may be buried, and some expense

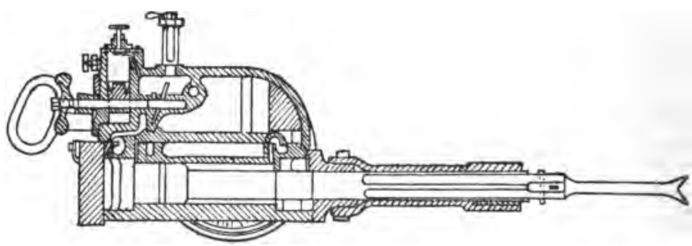


FIG. 414.—Longitudinal section of the Harrison coal-cutter.

has to be incurred in digging it out. It is also difficult to adjust the machine to cut in the dirt underneath the coal.

Coal-cutting machines have been chiefly used where the holing was in a bed of dirt about 18 inches above the bottom. There is a difficulty in moving heavy machines from one working place to another. These difficulties have made the profit of working coal-cutters in many cases so doubtful as to deter coal-mineowners from incurring the outlay of capital necessary for their use. In some cases where holing is done in the dirt by manual labour, the collier is able to pull out the fire-clay in large lumps instead of hacking it all to pieces; this saves a great deal of labour. The machine cannot do this, but has to cut the holing, whether it is dirt or coal, to dust.

Electrical Machines.—Great expectations were raised by the introduction of electricity, but hitherto electrical coal-cutting machines have made but small progress in Great Britain, although said to be experimentally successful in several cases.

The writer has heard it reported that they are extensively used in the United States of America, but has not had the opportunity of verifying the report. It is probable that, as a rule, the use of coal-cutting machines will be reserved for places where the holing is exceptionally hard, and for very thin seams of coal.

Since this chapter was first written, there has been a considerable development in experimentalization and in the practical adoption of holing-machines. Mr. W. E. Garforth, of Normanton, has perfected a disc coal-cutter of the Gillott and Copley type, which undercuts to a depth of 5 feet 6 inches. The improvements consist chiefly in the mode of attaching the cutters to the disc, so that they can be expeditiously changed when blunt, and in increasing the strength of every part by using cast and wrought steel. The disc is made in two parts, so that it can be taken along narrow gate roads. The machine will hole at the rate of 1 square yard a minute, the average speed for a whole shift, of course, being only a small fraction of that speed. One machine working in a 3-foot seam, holing partly in hard dirt and partly in coal, will undercut 1000 tons of coal in a week for many successive weeks. These machines are generally driven by compressed air, but Mr. Garforth has lately introduced electrically-driven machines.

Messrs. Clarke, Steavenson and Co., of Hoyland, near Barnsley, have also introduced an improved electrically-driven machine, also of the Gillott and Copley type, and have about sixteen of these at work. These machines are constructed of steel; perhaps no English firm has more experience of electrically-driven coal-cutters.

The Jeffrey Co., represented by John Davis and Son, Derby, has also brought out a similar electrically-driven machine, of which the distinguishing feature is that it is carried on one rail.

The Hurd machine is a bar cutter, similar to that described as Bower and Blackburn's (the original bar cutter was invented long before any of the modern machines). This machine has several ingenious arrangements, including a revolving head by which the position of the bar can be altered from its normal position, when cutting, at right angles to the rails, to a line parallel with the rails; it can also be laid horizontal or at an angle with a horizontal line. The bar is of great strength, fitted with cutters that can be quickly replaced when blunt. The machine is electrically driven. It can be made to cut as far under as required, say 5 feet 6 inches or 6 feet.

The rate at which the coal-cutters will work of course depends on the hardness of the material they cut and the power supplied.

The power actually required seems to vary from 15 H.P. to 30 H.P. at the machine, and the work done is in all cases about 1 square yard a minute, when the cutters are sharp and ground moderately hard, and the actual number of square yards holed in a nine-hour shift varies from 80 to 250.

At a rough estimate, it may be said that in the year 1899, of the coal got in Great Britain, a quantity not exceeding 2½ per cent. of the total is got with the aid of holing-machines, and in the United States perhaps 7 per cent. of the total quantity mined is undercut by machines.

CHAPTER XV.

TIMBERING: WOODEN AND STEEL BARS AND PROPS.

THE timber used varies with each locality according to the price of each particular kind of wood. On the eastern sea-board of England, Norwegian and Baltic fir are almost exclusively used; in the Midland and western counties this is used together with English timber, such as larch, oak, etc. In South Wales a good deal of pine of a heavy kind is imported from the south of France, which is used in addition to other kinds. English larch and French timber are generally used with the bark on, Baltic has generally but little bark. Where, owing to the nature of the ground, the bars will have to bear a heavy weight, round trees are used from 6 to 10 inches in diameter[1] (see Fig. 415); and where the weight is slight, the trees are often sawn down the middle, the flat side being placed against the roof (see Fig. 416). Square

FIG. 415.—Props and bars, round.

FIG. 416.—Props and bars, half-round.

timber is used sometimes, similar to railway sleepers. Steel bars are often substituted for wooden; they are sometimes supported by wooden props, and occasionally by steel props (see Fig. 417). They are often fixed in the stone sides of the road, and sometimes are carried by brick walls. The bars generally employed are girders, similar to those used for ordinary building purposes, as this form will sustain the greatest pressure for a given weight of steel. A rolled steel girder can be bent, but can hardly be broken; it is therefore a very safe kind of bar.

[1] Roughly squared hewn trees are also used.

But steel rails are liable to snap and fly under a breaking load, and it is therefore not so safe to use rails for bars. The steel is very much more costly, varying, according to the circumstances, from twice to six times the cost of the timber. But it is no more costly in labour to erect, and, when erected, will generally endure much longer. There are many places where steel bars once fixed would form a permanent support, because they will not decay; whilst, in the same place, a timber bar would rapidly decay, and within twelve months would have to be replaced, the labour of replacing being more costly than the wood replaced. In such cases there is a manifest economy in the use of steel. In roads which only serve a temporary purpose wood is to be preferred; and in roads where, owing to the excessive pressure or the unusual tenderness of the strata, bars, whether of timber or steel, will be rapidly broken or bent out of shape, timber is also probably the cheaper.

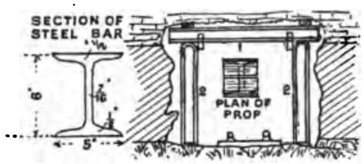


FIG. 417.—Props and bars, steel. 1, steel bars; 2, steel props with rounded ends.

The usual way of fixing a bar on to props is shown in Fig. 415. The diameter of bars used for the timbering in mines in this country varies from about 3 up to 12 inches; occasionally, no doubt, bars of greater diameter are used. The timber for props also varies in this country from about 3 inches to about 10 or 12 inches in diameter. In some mines, however, which the writer has visited, much thicker timber is used; for instance, in the Calumet and Hecla Mine, on Lake Superior, timber props are set 3 feet in diameter.

In order that a bar may not be easily knocked off the prop, it is generally tightened by a wedge at the top; sometimes the end of the bar is notched and the prop also, as shown in Fig. 418. In other cases an iron spike is simply knocked into the bar in front


FIG. 418.—Props and bars, notched.

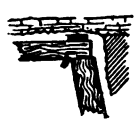


FIG. 419.—Props and bars, spiked.

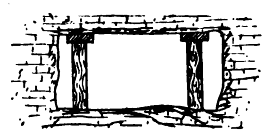


FIG. 420.—Props in heading.

of the prop, as shown in Fig. 419. In many cases props only are used, as shown in Fig. 420; in other cases props are dispensed with, as shown in Fig. 421. Fig. 422 shows a cross-section and longitudinal section of a timbered roadway. It will be seen that over the bars are placed poles or planks, going longitudinally, and over these the space is filled up with dirt, so that there may be no cavity above the timber in which gas can accumulate; and the

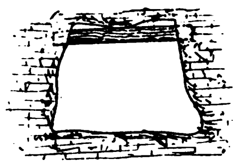


FIG. 421.—Bars in heading.

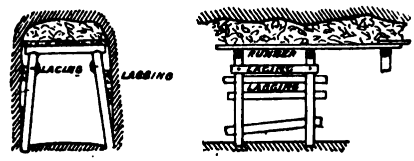


FIG. 422.—Sections of timbered heading.

roof above is also held securely in position. Lagging is sometimes laid at the sides of the props to prevent pieces falling off between. At the working face the collier must protect himself with sprags under the coal, and with props and sprags against the front of the coal, and with chocks, as shown in Fig. 423. The

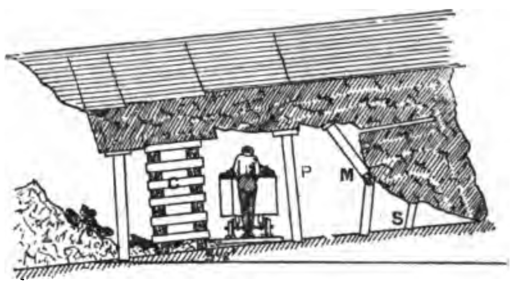


FIG. 423.—Timbering working face. P, props; C, cog; M, cockermeg; S, sprag.

reader is referred to an excellent book by Mr. A. R. Sawyer, on "Accidents in Mines," of which the writer has made some use in preparing this chapter.

Props must be fixed nearly at right angles to the inclination of the seam, but in steep seams the props should be inclined a little to the rise of a line at right angles to the dip, say 2° or 3°, as is shown in Fig. 424; because the tendency of the roof is to move towards the dip, and if the prop were fixed originally at right angles, the movement of the roof might loosen it, but if it is fixed slightly inclined towards the vertical, the movement of the roof towards the dip tightens the prop. The prop is tightened with a wedge at the top driven in with a hammer; this wedge is sometimes called the lid. If of small dimensions, it merely serves to tighten the prop; but if, instead of a wedge, a strong cross-piece 2 feet long is fixed, and that again is tightened with a wedge, the prop, by means of this cross-piece, supports a larger area of roof (see Fig. 425). The roof should be so supported by props and lids that any unseen joints may be covered; otherwise accidents occur through stones falling out along lines of fault or other joints which could hardly be observed. Instead of, or rather in addition to, props, cogs or chocks are often placed. These consist of pillars of timber made of cross-pieces, each about 2 feet long and 5 inches square, of hard wood (see Fig. 423). These may be pulled down and rebuilt. Cast-iron props, as shown in Fig. 426, have often been used; in some places, wrought-iron or steel props are used.

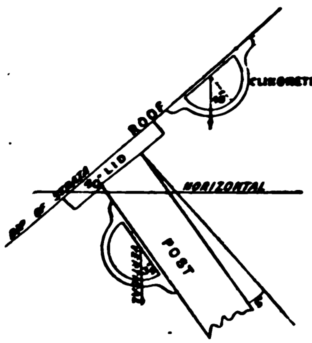


FIG. 424.—Prop and angle measure.

In the working places props have frequently to be withdrawn, and, to avoid accidents, this must be done with great care. If the prop is very fast, a temporary prop may be erected beside it

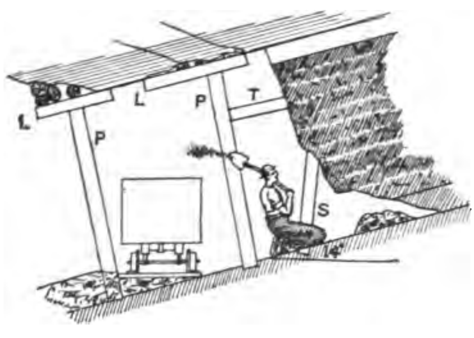


FIG. 425.—Props and large lid. P, props; L, lids; S, sprag; T, strut.

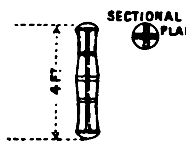


FIG. 426.—Cast-iron prop.

to sustain the roof while the workman is cutting loose the other prop; then both the props may be withdrawn by ropes or chains, the workman going to a safe distance. A method of drawing the props commonly used is by means of a lever and chain (Fig. 427). The chain is fastened round the prop to be withdrawn, and the lever rests against a sound prop. In some cases a rope (see Fig. 428) is made fast to the prop to be drawn and to several sound props. Then the workmen, pulling the rope sideways, throw a great strain on the loose prop, and so cause it to be withdrawn. A prop that is fast may be loosed by means of a tool called a jobber (see Fig. 429), with a handle about 7 feet long.

Many colliery managers consider that props should be withdrawn, not merely for the sake of saving the timber for future use, but to facilitate the regular subsidence of the strata behind the

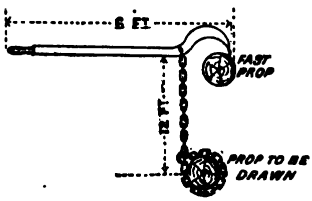


FIG. 427.—Ringer and chains and tools.

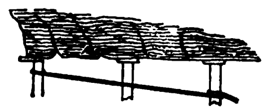


FIG. 428.—Rope and props.

face, and in this way diminish the likelihood of a weight coming on to the working place.

When a bar across a road has been broken or bent by the weight, if it were withdrawn the roof would fall; therefore additional bars or liners must be placed on each side of the broken bar before it is withdrawn. In this way the workmen are protected and the roof is kept up. When the roof has crushed down the bars for some considerable length, and it is necessary to restore the road to its original height, if the bars are drawn, a great deal of loose shale may fall, and it may become an expensive

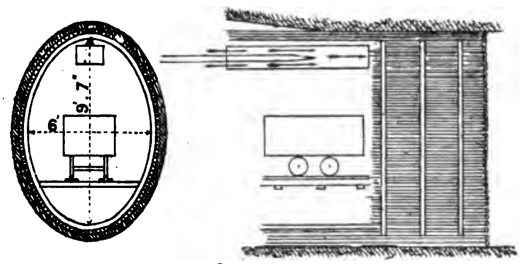
FIG. 429.—Jobber.

and dangerous job to re-timber the road. In such a case the writer has successfully used hydraulic jacks to raise the bars, subsequently setting new props and bars without letting the roof fall. Screw-jacks have also been used for the same purpose.

The need for careful propping is shown by the accidents occasionally happening for want of props, either because the unsupported area was too large, or because of joints, faults, and "pot-lids." Accidents have also happened owing to want of sufficiently careful calculation as to the weight carried by bars at a place where two main roads cross. In this case the bars of one road have to be carried on two bars forming a bridge across the roads at right angles. The weight on these bridge bars may be excessive. Great care has to be taken not only that they are sufficiently strong at the beginning, but that, in changing them, sufficient temporary props are fixed under the bars of the other road resting upon them.

When a heading is driven in quicksand, piles are driven in advance in a horizontal direction, similar to the method used in sinking shown in Fig. 189. It is not often that this has to be done, but it is not unfrequently the case that in driving through soft ground boards have to be driven in diagonally on the roof and sides, to prevent the earth from running into the excavation.

Figs. 430 and 431 are from sketches made by the writer in Saxony; they show in plan and elevation a heading in a thick and



THE IRON RINGS ARE Ω SECTION & 20 LBS. TO THE YARD

FIGS. 430, 431.—Sections of heading in coal, with iron rings and wood poling.

tender seam of coal. Wrought-iron rings are placed at intervals of about 2 feet. Between these rings and the coal very small polings of fir are placed, forming a complete lining of timber.

Another method of timbering is shown in Fig. 432, a sketch by the writer at a colliery in the north of France. In this case iron bars bent in the form of a horseshoe, weighing about 25 lbs. to the yard, are placed at intervals of about 2 feet. These bars are in section shaped like a girder weighing 25 lbs. to the yard. The space between them is filled up with $1\frac{1}{4}$-inch planks, which fit against the webs of the iron bars between the flanges, making a smooth and strong lining for the roadway, which is 7 feet wide.

Timbering is liable to be destroyed in the following ways: First, by decay; in warm and dry mines this is very rapid, the timber being sometimes destroyed in a few months. It is said that this decay is arrested by damping the air with spray-jets. In many places, especially in very wet places, the timber will sometimes endure for generations.

Bars are often broken by a combination of pressure on the centre of the bar from the roof, and on the ends of the bar from the side. In order to prevent this end pressure from breaking the timber, care must be taken that the bars are kept free at the ends; this in some mines involves continual cutting away of the shale which squeezes in.

Rules for Strength.—The strength of a bar is in direct proportion to its width; thus a bar 12 inches wide is twice as strong as a bar 6 inches wide of the same depth and length. The strength of a rectangular beam is in proportion to the square of the depth. Thus, if there are two bars of equal width, one 6 inches deep and the other 12 inches deep, their proportional strength is as $6^2 : 12^2 :: 36 : 144$; or as 1 to 4. The longer the bar the weaker it is; thus a 12-foot bar is only half as strong as a 6-foot bar, other measurements being equal. Thus if a bar is 6 inches wide and 12 inches deep by 8 feet long between supports, it will bear twice the weight in the centre that a bar 12 inches wide and 6 inches deep and 8 feet long would bear, and twice the load that a bar 6 inches wide, 12 inches deep, and 16 feet long between supports would bear.

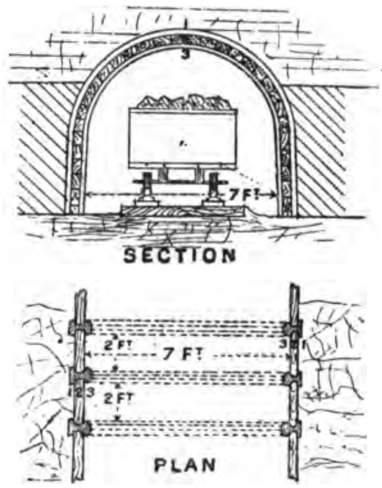


FIG. 432.—Method of supporting a roadway. 1, foot of rail; 2, timbering between rails; 3, head of rail.

A bar supported at both ends will bear twice the load in the centre that it will bear if only supported at one end. If the load is distributed equally all along the bar, it will carry twice the weight that it will bear if the load is all in the centre. If the two ends of a bar are very firmly built in for a length of at least 18 inches and held tight (on the top and bottom), it will bear half as much again as it would if the ends were loose with distributed load.

The strength of beams is given in "Molesworth's Pocket-Book" by the following rule. Rectangular beams—

W = breaking weight at centre of beam supported at both ends, in hundredweights.
B = breadth of beam in inches.
D = depth ,, ,,
L = length ,, ,,
K = coefficient, varying with timber used.

K = for yellow pine, 10; for red pine, 13; Memel, 12; spruce, 12; English oak, 15.

(N.B.—These strengths as given in Molesworth do not agree with those given by Unwin, and they must be regarded by the student rather as indications of differences in strength than as correctly representing the exact relative value of various timbers.)

Rule.—$W = \frac{4KBD^2}{L}$

Example.—Let B = 6, D = 6 L = 72 (6 feet), and K = 10; then—

$$W = \frac{4KBD^2}{L} = \frac{4 \times 10 \times 6 \times 6^2}{72} = 120 \text{ cwt., or } 6 \text{ tons}$$

This rule may be simplified when K = 10, as follows:—

W = breaking weight in tons; then—

$$W = \frac{2BD^2}{L}$$

This is easy to remember, as most of the figures are 6's. Thus, beam 6 inches wide, 6 inches deep, 6 feet long; breaking weight, 6 tons.

Rule for strength of round beams is given by Molesworth as follows:—

$W = \frac{4KV}{L}$

$V = 4{\cdot}7R^3$.

R = the radius, or half-diameter, of the log in inches.

The following example shows how the rule is worked:—

Let K = 10, as in the foregoing example; let the diameter be 8 inches, the radius 4 inches, and L be 72 inches; then—

$$W = \frac{4 \times 10 \times 4{\cdot}7 \times 4^3}{72} \quad 167{\cdot}1 \text{ cwt.}$$

or, roughly speaking, an 8-inch bar 6 feet between the supports will require a load of 8 tons in the centre to break it.

It will be seen that this rule agrees approximately with some results obtained by testing mining timber with the 100-ton testing-machine at the Yorkshire College. The tests were made by Professor Goodman and the writer upon bars of ordinary mining timber in their ordinary condition before use, as taken from a pile out of doors. The results are shown in the following table. A simple rule for the breaking weight of round beams is given in the last column.

TABLE XIV.—TIMBER TESTS.

The following were tested as beams, and were loaded in the centre on a flat iron saddle 10 inches long, the ends being free and supported on 3-inch semicircular blocks, smooth and free to turn, the span being measured from centre to centre of these blocks. The test-pieces consisted of round timber bars neither hewn nor sawn, but just as grown and cross-cut, the oak without bark.

Description of timber.	Span.	Girth at centre.	Diameter.	Actual breaking load.	Corrected span.	Calculated breaking load.	Remarks.
	Feet.	Feet.	Inches.	Tons.	Inches.	Tons. $W = \frac{1 \times D^3}{L}$	
English—							
Larch ...	5	1·94	7·4	10·64	50	8·10	The seventh column is calculated by the rule $W = \frac{D^3}{L}$ Here W is the breaking load in the centre of the beam with ends loose in tons, D is the diameter in inches, L is the span in inches. In the above experiments the load was not entirely in the centre, because the load was placed on an iron block 10 inches long. So in the sixth column of the table the beam is taken as 10 inches shorter in each case, which reduced length is called "corrected span," and is used in the seventh column.
" ...	5	1·94	7·4	8·76	50	8·10	
Oak ...	5	1·66	6·3	6·1	50	5·00	
" ...	5	1·58	6·0	5·8	50	4·32	
Norwegian—							
Fir ...	5	2·07 (no bark)	7·9	10·22	50	9·86	
" ...	5	2·00 (no bark)	7·6	8·67	50	8·77	
" ...	5	2·12	8·1	11·80	50	10·62	
" ...	5	1·55 (no bark)	5·9	5·30	50	4·10	
" ...	5	1·57 (no bark)	6·0	5·23	50	4·32	
" ...	5	1·68	6·4	5·12	50	5·24	
" ...	4 ft. 6 in.	1·77	6·8	7·82	44	7·14	
" ...	4 ft. 6 in.	2·00	7·6	10·15	44	9·97	
" ...	4	1·24	4·7	4·21	38	2·73	
" ...	4	1·34	5·1	4·60	38	3·49	
" ...	4	1·27	4·9	5·27	38	3·09	
" ...	3 ft. 6 in.	1·19	4·5	4·36	32	2·84	
" ...	3 ft. 6 in.	1·19	4·5	4·36	32	2·84	
" ...	3 ft. 6 in.	1·13	4·3	3·72	32	2·48	
English—							
Larch ...	3 ft. 6 in.	1·10 (⅛ in. bark)	4·2	3·68	32	2·31	
" ...	3 ft. 6 in.	1·14 (⅛ in. bark)	4·4	3·17	32	2·66	
Ash ...	3 ft. 6 in.	0·97	3·7	2·92	32	1·58	

The strength of rolled steel girders of H section (see section, Fig. 417) may be calculated by the following rule :—

W = breaking weight in tons in centre of span.
x = tensile strength of material in tons per square inch.
A = area of one flange (either top or bottom) in square inches.
D = depth of girder from top to bottom in inches.
L = span in inches.

$$\text{Then } W = \frac{4x \times A \times D}{L}$$

For rolled steel joists, $x = 25$ tons,[1] and $4x = 100$ tons. The safe load in structures is $\frac{1}{5}$ of the breaking load, or for rolled steel girders the safe load is say 5 tons per square inch. Therefore the formula for a safe load for a girder supported at both ends, and all the weight in the centre, and ends free, would be—

$$W = \frac{20 \times A \times D}{L}$$

To take an example: Let A = 5 inches $\times \frac{1}{2} = 2\frac{1}{2}$ square inches, let D = 6 inches, and L = 72.

$$\text{Then the breaking load} = \frac{100 \times 2\frac{1}{2} \times 6}{72} = 20{\cdot}83 \text{ tons}$$

$$\text{And the safe load} = \frac{20 \times 2\frac{1}{2} \times 6}{72} = 4{\cdot}16$$

If the load were distributed evenly throughout, the girder would safely sustain double the above load, or 8 tons; and if the ends were built in very firmly to a length of at least 18 inches, with a heavy load on them, and the load distributed, it would bear 50 per cent. more, or 12 tons.

The following table gives the breaking load and safe load of rolled steel H girders :—

TABLE XV.—STEEL GIRDERS AS BARS.

Description of joist in list.	Area of flange.	Depth of girder from top to bottom.	Calculated breaking load (tons), load in centre, ends free, for a span of				Calculated safe load (tons) for a permanent structure, load in centre, ends free, for a span of			
			70 in.	100 in.	150 in.	200 in.	70 in.	100 in.	150 in.	200 in.
Depth. Width.	sq.in.	in.								
3 in. × 3 in.	0·75	3	3·21	2·25	1·50	1·12	0·64	0·45	0·30	0·22
5 in. × 3 in.	0·90	5	6·43	4·50	3·00	2·25	1·28	0·90	0·60	0·45
6 in. × 5 in.	2·50	6	21·42	15·00	10·00	7·50	4·28	3·00	2·00	1·50
8 in. × 6 in.	3·00	8	34·28	24·00	16·00	12·00	6·85	4·80	3·20	2·40
10 in. × 6 in.	3·54	10	50·57	35·40	23·60	17·70	10·11	7·08	4·72	3·54

[1] 28 to 30 is, perhaps, nearer the figure, but 25 tons is within the mark. For wrought iron, $x = 20$ tons.

In order to determine the strength of timber props, the writer procured a number of specimens of oak, larch, and Norway props of the size and condition commonly used in mines. These were put in the testing-machine, and subjected to an end pressure till they crushed. The results are shown in the table on the following page.

It will be noted that there does not appear to be any serious difference in the strength of the three kinds of wood as tested but the oak props were not so straight as the larch, and the larch was not so straight as the Norway. It will also be noted that the 7-feet props crushed with a less weight than the 4-feet props, and these again with a less weight than the 2-feet props. The weight that would be required to crush a prop longer than 7 feet can be ascertained roughly from the above tests by calculation. Assuming that the long prop was quite straight, it is probable that the crushing load will not fall much below 1 ton per square inch, if the length does not exceed twelve times the diameter.

Professor Unwin, in his book on "The Testing of Materials of Construction," quotes experiments by Professor Lanza, according to which the crushing strength of posts 7 to 10 inches in diameter, 12 feet long, and 2 feet long, of yellow pine, was very nearly 2 tons per square inch. It is probable that this was an excellent quality of seasoned timber. Professor Lanza also made tests of posts up to 30 feet in length, from which it appears that a yellow pine post, of which the length is fifteen times the diameter, requires a crushing load of $1\frac{3}{4}$ tons per square inch; and with a length from thirty to forty times the diameter, the crushing load is $1\frac{1}{3}$ ton per square inch; and with a length fifty to sixty times the diameter, the crushing load is $\frac{2}{3}$ ton per square inch. With white pine the strength is less. With a length ten times the diameter, the crushing load is $1\frac{1}{9}$ ton per square inch; and with lengths from ten to thirty-five times the diameter, the crushing load is about $\frac{9}{10}$ ton; while with lengths forty-five to sixty times the diameter, the crushing load is about 0·44 ton per square inch. The permanent or safe load is, of course, much less than the loads above given in all cases.

It would seem, from the various observations above recorded, that the strength of a straight prop or strut is in some measure proportional to the relation between length and diameter, and that a prop 12 inches in diameter and 12 feet long will bear approximately as great a load per square inch as a 6-inch prop 6 feet long. The relation of the ratio of the length and diameter of props and the crushing load per square inch, as worked out from the Yorkshire College tests, is given in Table XVII.

TABLE XVI.—PARTICULARS OF TIMBER TESTS.

The test-pieces consisted of round timber props, neither hewn nor sawn, but just as grown and cross-cut, the ends not squared, the oak without bark.

The following were tested as struts as follows: at the top and bottom of each prop a piece of board about ½ inch thick was placed, to give a level bedding, the compression was applied through flat and level iron plates at the top and bottom, the prop being vertical.

Description of timber.	Length.	Girth in feet. Centre.	A end.	B end.	Crushing load.	Sectional area.	Crushing load per sq. in.	Remarks.
	ft. in.				tons.	sq. in.	tons.	
Oak	7 0¼	1·55	1·41	1·67	32·90	27·33	1·20	Crushed at A end.
,, ...	7 0¼	1·79	1·79	1·88	50·00	38·48	1·29	Failed in middle.
,, ...	3 6¼	1·66	1·75	1·66	44·01	32·17	1·37	Failed between middle and B.
,, ...	3 6	1·63	1·71	1·64	42·25	31·17	1·35	Crushed at middle.
,, ...	3 6	1·57	1·62	1·49	40·15	28·27	1·42	Crushed at A end.
,, ...	3 5¼	1·66	1·62	1·75	36·51	32·17	1·13	Failed in middle.
			thin	bark				
Ash	3 0¼	1·00	0·94	1·00	17·14	10·75	1·59	,, ,,
			thin	bark				
,, ...	2 8	0·96	1·00	0·92	18·29	10·75	1·70	,, ,,
			thin	bark				
,, ...	2 7	0·8	0·78	0·83	8·67	7·54	1·14	,, ,,
Larch ...	7 0¼	1·68	1·84	1·62	31·71	33·18	0·95	Failed by bending; bent and restraightened.
,, ...	7 0	1·59	1·52	1·75	31·17	30·19	1·03	Failed between middle and A.
,, ...	7 0	1·93	2·06	1·83	50·00	43·00	1·16	Load on some minutes; prop gradually failed.
,, ...	6 11¾	1·60	1·54	1·67	24·61	29·22	0·84	Slight initial bend.
,, ...	6 11½	2·15	1·92	1·78	45·33	44·17	1·03	Failed between middle and B.
,, ...	4 0¼	1·29	1·22	1·17	20·66	17·34	1·19	Crushed at A end.
,, ...	4 0	1·19	1·10	1·20	20·79	15·20	1·37	Failed by bending; bent and restraightened.
,, ...	3 6	1·97	1·97	1·90	46·95	43·00	1·09	Crushed at B end.
,, ...	3 6	1·96	1·99	1·91	59·04	44·17	1·34	Failed in middle.
,, ...	3 6	1·67	1·65	1·64	43·47	31·17	1·39	Failed in bending; bent and restraightened.
,, ...	3 5¼	1·93	2·03	1·84	41·10	43·00	0·95	Failed in middle.
,, ...	2 0¼	1·09	1·06	1·09	21·57	13·20	1·63	,, ,,
,, ...	2 0¼	1·15	1·13	1·20	26·15	15·20	1·72	Failed at A end.
,, ...	2 0	1·13	1·09	1·15	24·58	14·52	1·69	Failed in middle.
		no	bark					
Norweg. fir	6 1¾	1·57	1·53	1·66	40·72	28·27	1·44	Wet; failed at A end.
,, ...	6 0	1·64	1·76	1·60	35·57	32·16	1·10	,, ,,
,, ...	5 11¼	1·62	1·56	1·66	39·40	30·19	1·30	,, ,,
,, ...	3 7¼	1·03	0·97	1·10	17·77	11·94	1·49	Failed at B end.
,, ...	3 7	1·07	1·11	1·03	16·08	13·20	1·22	,, ,,
,, ...	3 6	1·11	1·15	1·06	17·66	13·85	1·27	,, ,,
,, ...	3 0	1·22	1·24	1·19	25·22	16·61	1·52	,, ,,
			no bark					
,, ...	2 11¼	1·18	1·20	1·14	18·24	15·90	1·14	,, ,,
			no bark					
,, ...	2 11¼	1·22	1·25	1·18	16·75	16·61	1·01	,,
,, ...	2 11½	1·17	1·20	1·18	23·30	15·90	1·46	,, ,,
,, ...	2 1¾	1·25	1·28	1·23	30·13	18·09	1·66	Cut from same piece. Failed in middle.
		no	bark					
,, ...	2 1½	1·29	1·26	1·30	33·4	18·85	1·77	Cut from same piece. Failed at B end.
		no	bark					
,, ...	2 1¾	1·29	1·31	1·26	31·38	18·85	1·66	Cut from same piece; failed at A end.
		no	bark					
,, ...	2 1¼	1·25	1·23	1·26	31·70	18·09	1·75	Cut from same piece; failed at A end.

TABLE XVII.—TIMBER PROPS.

Table showing average crushing load per square inch for each kind of wood and each length, and also the ratio of the length to the diameter of the average of the samples of each kind and length.

Description of timber.	7 feet.		6 feet.		4 feet.		3½ feet.		3 feet.		2 feet.	
	Average crushing load per square inch in tons.	Length in inches divided by average diameter in inches.	Average crushing load, etc.	Length in inches divided by diameter, etc.	Average crushing load, etc.	Length in inches divided by diameter, etc.	Average crushing load, etc.	Length in inches divided by diameter, etc.	Average crushing load, etc.	Length in inches divided by diameter, etc.	Average crushing load, etc.	Length in inches divided by diameter, etc.
Eng. oak	1·24	13·02	—	—	—	—	1·31	6·6	—	—	—	—
,, ash	—	—	—	—	—	—	—	—	1·47	10·28	—	—
,, larch	1·00	12·40	—	—	1·28	10·5	1·19	5·80	—	—	1·68	5·6
Norweg. fir ...	—	—	1·28	11·6	—	—	1·32	10·30	1·28	7·9	1·71	4·90

Props made of steel girders with part of the web cut out at each end, and the flanges bent over (Firth's patent), are a good deal used. The crushing load may be calculated from the following formula, and the table shows the crushing load that may be put upon props of the section given. It must in all cases be borne in mind that the permanent safe load is about one-fifth of the load that will immediately crush a prop.

To find the buckling load for a steel prop.

Rule.—

$$B = \frac{L}{1 + a \times r^2}$$

$B \times A$ = total buckling load for prop.

B = buckling load in tons per square inch.

A = sectional area of metal in square inches.

L = elastic limit of material.

a = a constant depending on section and material.

$$r = \frac{m}{d}$$

When neither end is rigidly fixed, $m = l$, d = least width in inches, l = length in inches.

Example.—Prop is 4 feet long, of H-section rolled girder, mild steel, 8 inches × 6 inches. Sectional area is 8·43 square inches.

L = 16 tons for wrought iron or mild steel.

$a = \frac{1}{1200}$ for rolled joists H section.

$m = l$ = length in inches = 48.

d = 6 inches.

A = 8·43 square inches.

$$\text{Then } B = \frac{16}{1 + \left\{\frac{1}{1200} \times \left(\frac{48}{6}\right)^2\right\}} = \frac{16}{1 + \left\{\frac{1}{1200} \times \frac{48 \times 48}{6 \times 6}\right\}} = 15{\cdot}18 \text{ tons}$$

$$\text{And } B \times A = 15{\cdot}18 \times 8{\cdot}43 = 127{\cdot}96 \text{ tons}$$

TABLE XVIII.

Calculated by the rule above given, for H-section rolled joists or girders used as props, both ends free.

Description.	Thickness of web.	Average thickness of flange.	Total sectional area.	Buckling load for a strut of which the ends are not fixed,[1] for a span of				Safe permanent load on the same struts. Factor of safety = 5.			
dep. wid.	in.	in.	sq. in.	4 ft.	6 ft.	8 ft.	12 ft.	4 ft.	6 ft.	8 ft.	12 ft.
in. in.											
3 × 3	0·18	0·25	1·9	25·04	20·53	16·40	10·39	5·01	4·10	3·28	2·08
5½ × 4½	0·37	0·5	6·00	87·66	79·08	69·60	51·78	17·53	15·82	13·92	10·36
6 × 5	0·44	0·5	7·11	105·58	96·98	86·95	67·26	21·12	19·39	17·39	13·45
8 × 6	0·36	0·5	8·43	127·96	120·38	111·10	91·12	25·59	24·08	22·22	18·22

[1] A prop as set up between roof and floor is a "strut the ends of which are not fixed."

Safe Loads.—It must be borne in mind that the safe load of any material is much less than the crushing load. For instance, whilst the bar 6 inches square, 6 feet between the supports, will carry 6 tons in the centre, a load of 1 ton would be sufficient for a permanent loading.

In timbering, the factor of safety commonly employed in building is 7, that is to say, that if a load of 7 tons will break a bar, a load of only 1 ton should be put on it permanently. It is, perhaps, unnecessary in mines to allow such a large factor of safety, because the timbering has seldom to remain permanently; and it is probable that a factor of safety of 2 or 3 is sufficient both for bars and props in a mine—that is to say, that if the calculation shows that a bar would break with a load of 6 tons, it may probably be safely required to carry a load of 2 to 3 tons. And the same with regard to props. If the ultimate crushing strain of a prop is say 40 tons, it is probable that it may safely carry a load of 20 tons, because it will only have to bear the load for a short time. Of course, in addition to this, allowances must be made for deterioration of the timber by decay, or for weakening of the timber by cutting portions of it away.

CHAPTER XVI.

UNDERGROUND HAULAGE: JIGS, SINGLE ROPE, TAIL ROPE, ENDLESS ROPE, CHAIN, LOCOMOTIVES, COMPRESSED AIR, ELECTRIC, ROPEWAYS ON SURFACE.

THE underground conveyance of mineral is a matter of continually increasing importance, not merely because of the rapidly increasing production, but because, as the mines get deeper, the area of mineral got from one pair of shafts increases. Underground haulage roads are now not unfrequently two miles in length, and sometimes reach a length of four miles. The total length of underground railway in Great Britain, is equal to half the length of surface railway, but the extent of underground railway is increasing faster than that of surface railways.

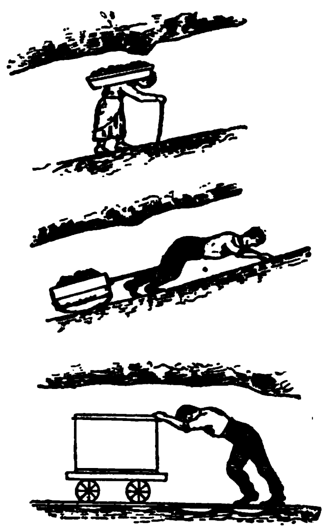


FIG. 433.—Methods of hurrying.

Various means of haulage are shown in Fig. 433. The most primitive method of conveyance is that of putting the mineral into a bag or basket, and carrying it on the head or shoulders, a plan which is still adopted in some parts of the hæmatite iron-mines in the Forest of Dean, and in many places abroad.

In some mines the mineral is conveyed in wheelbarrows; this method was pursued at the Mona Copper-mines in Anglesea. Iron rails could not be laid down because of the corrosive action of the water, which would have rapidly destroyed them, and therefore wooden planks were laid in the centre of the road, along which the wheelbarrows were run.

Another method is to employ a basket or box, with slides like sledge-runners underneath. This is suitable for the conveyance of minerals down steep inclines for short distances, and is often employed in thin seams of coal to convey the coal down the branch gate to the waggon-way, where they are emptied into the waggon or into a heap beside the waggon-way.

The most usual plan is to put the box or basket on to four wheels, which run on rails. The old-fashioned kind, still largely used, has wheels with thin edges; the rail is shaped like an angle-iron, one flat side laid on the ground, the other standing inside, thus forming a flange to guide the wheels. A much superior plan is to put the flange on to a broad wheel, like a railway wheel, which runs along a round-topped rail. The diameter of the wheel varies with the size of the waggon from 6 inches to 2 feet. They are generally now made of cast steel, which is much lighter and more durable than cast iron. Steel rails are also now used, because they are more durable than wrought iron.

The sections of rails commonly used, as shown in Fig. 434, are flat-bottomed, bridge, and double-headed; flat-bottomed are generally preferred, the bridge rails being rather more difficult

Fig. 434.

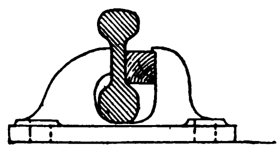


Fig. 435.

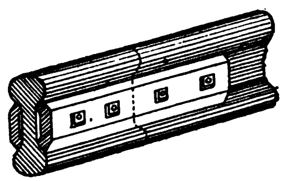


Fig. 436

FIG. 434.—Sections of rails. FIG. 435.—Double-headed rail in chair.
FIG. 436.—Double-headed rail in fish-joints.

to bend round corners. These rails are spiked to the sleeper. Where double-headed rails are used, a chair is necessary (see Fig. 435). To make a first-class road, the rails should be fish-jointed (see Fig. 436) to prevent the ends jumping up, or else a joint-chair must be used (see Fig. 437). Wrought-iron or steel sleepers are often used in place of wooden sleepers. Where these are used, the rail is fastened by being slipped under projecting clips on the sleeper, sometimes afterwards tightened with a wedge. Rails as light as 10 lbs. to the yard are sometimes used where the boxes only contain 3 or 4 cwt. of coal; 16 to 18 lb. rails are used for boxes containing 7 or 8 cwt. of coal; for main roads and boxes containing 10 cwt. of coal, rails 25 and 35 lbs. to the yard are often used.

Fig. 438 shows a set of cast-iron points and crossings. The points are fixed, and the waggons must be turned off by hand; but this kind of turn-out is frequently only used for waggons coming one way. Fig. 439 shows a similar arrangement where loose points are used. Fig. 440 shows a long centre rail moved across the road instead of ordinary points. Fig. 441 shows a double set of points and crossings similar to those on a railway.

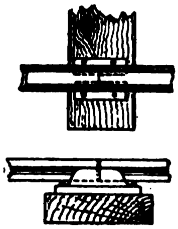


FIG. 437.—Joint-chair.

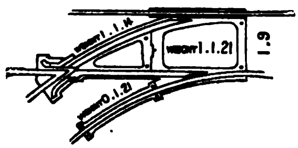


FIG. 438.—Cast-iron points and crossings: left-hand turn.

Fig. 442 shows A, cast-iron crossing; B, a turn and crossing; C, grooved plate for pit-bottom; D, cast-iron check rail.

Pit Waggons.—These are called indifferently waggons, corves, tubs, boxes, or carts (see Fig. 443). They are generally made of wood, bound with iron at the top and corners, with iron or steel draw-bar, iron hoops to the buffers; but they are sometimes made entirely of iron or steel, but have generally wooden sole-pieces. In the north of England iron waggons seem to have the preference.

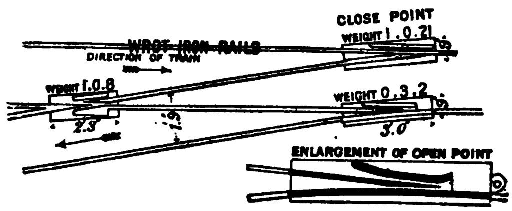


FIG. 439.—Cast-iron points and crossings.

The thinner the material of which the waggon is made, the greater the weight it will hold for given external dimensions. The axles are generally made of steel; the wheels are sometimes placed between the sole-pieces, but more often outside the sole-pieces. They are sometimes loose on the axles, revolving between collars. This arrangement reduces the friction to a minimum, because, as the road is seldom perfect, and there are always a great number of curves, one wheel must go faster than the other at many places. On the other hand, unless the wheel is a very good fit on the axle and between the collars, it may wobble and miss the points. For this reason the wheels are generally fast on

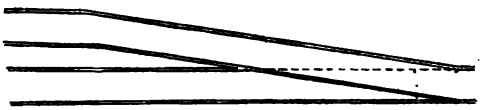


Fig. 440.

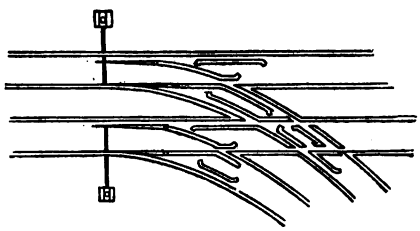


Fig. 441.

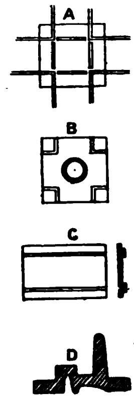


Fig. 442.

FIG. 440.—Swing point and crossing. FIG. 441.—Double set of points and crossings, wrought iron. FIG. 442.—Cast-iron plates.

the axles, like those of railway waggons, the axles revolving on pedestals or caps.

In some cases the waggons are made entirely of iron bars, forming a framework in which lumps of coal are placed, and

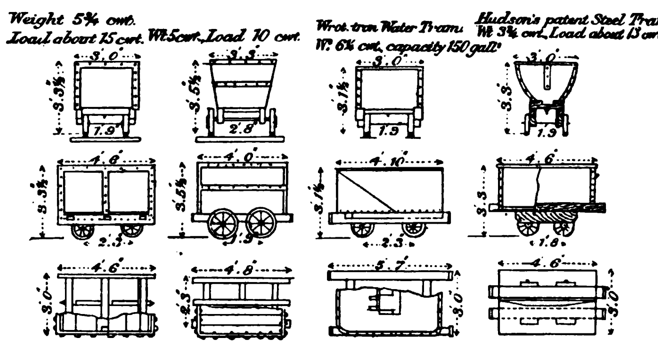


FIG. 443.—Diagrams of pit trams.

through which any slack made in transit falls. This plan is not now in favour, because of the amount of coal-dust distributed along the roads.

In some parts of the Midland Counties the sides of the box are very shallow, say only 6 inches. As coal is loaded above the side, rings or garlands of wood or iron are placed round the coal, four or five rings being placed sometimes one above the other, and the coal is built up to a height of 6 feet above the ground. This facilitates unloading on the bank.

Lubrication.—A good deal of attention has been given to the lubrication of the axles, which is essential for economical haulage. The lubrication is often effected by means of some grease applied with a stick to the under side of the axle, or oil is used poured from a can. In order to apply the lubricant, it is sometimes necessary to turn the waggon upside down, and the lubricant is often applied when the waggon is upset over the screens for the purpose of throwing out the mineral. In some cases the waggon is stopped with the axles just opposite four squirts, from which little jets of oil are sent by a force-pump.

Another plan is shown in Fig. 444. Underneath the road is a tank containing oil; revolving in the oil are two wheels, the axles of which are carried on springs. The waggon is run over these wheels, which touch against the waggon axle; the wheels yielding as the axle passes over, the oil on the wheel is communicated to the axle. Where this system of lubrication is employed, the pedestal is only on the upper side of the axle; a collar to prevent the box from being lifted off the wheels is fixed a little on one side of the pedestal, so that the lubricating wheel comes against the axle. These lubricators are fixed on the pit-bank and at the pit-bottom and at intervals throughout the mine, so that the waggons are oiled perhaps four times each journey. Sometimes the axle is lubricated by grease placed in bottles fixed in the sole-pieces, running out through a small hole partly filled with a pin, as in the needle oil-cup. For the success of this device, it is necessary to have good workmanship for all the parts, and a uniform quality of oil.

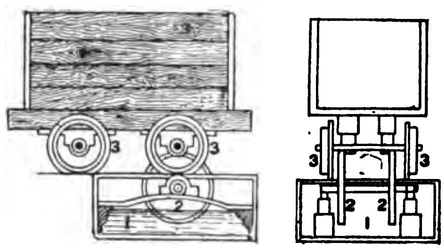


FIG. 444.—Tram axle-greaser. 1, oil-box; 2, oil-wheels rubbing against axle of tub; 3, wheels and axles oiled when passing over oil-well.

Hurriers and Horses.—When the waggon has been loaded, it is often taken along the branch gates, particularly in thin seams, by a lad called a hurrier, who will push it at a great pace if the gradient is in his favour. On the main roads or where the height is sufficient, a pony or horse is employed. A lad with a small horse will take ten or twenty times the weight that he can without the horse. Horse-haulage is very convenient and economical in all mines where the roads are level or where a moderate gradient is in favour of the load. But the most economical gradient for horse-haulage is where the work done by the horse is equal going out with the loaded corf or coming in with the empty.

This gradient may be calculated by the following rule :—

Let G = fraction representing gradient.
F = fraction representing friction.
L = total weight of loaded train.
E = total weight of empty train.

$$G = F \times \frac{L - E}{L + E}$$

Let $F = \frac{1}{80}$, and L = 10 tubs each 14 cwt. + 1 lad = 1 cwt. + $\frac{1}{2}$ horse = 3 cwt. = 144 cwt. E = 10 tubs each 4 cwt. + 10 cwt. of timber + lad and $\frac{1}{2}$ horse = 4 cwt. = 54.

Then, applying the rule—

$$G = \frac{1}{80} \times \frac{144 - 54}{144 + 54} = \frac{1}{176}$$

that is, the gradient is 1 in 176—that is to say, $\frac{1}{5}$ inch per yard, the loaded train going downhill and the empty train going uphill.

Jigs.—Where the incline is steep, the loaded tub going downhill, no horse is required, as the weight of mineral descending is sufficient to pull the empty waggon up the hill. A road so arranged is called a jig, or self-acting incline. The gradient required for this may be calculated from the following rule :—

G = fraction representing gradient.
L, E, F = same as before.
K = 120 per cent., or $\frac{6}{5}$.

$$\text{Then } G = \frac{L + E + \frac{L + E}{5}}{L - E} \times F \times K$$

Let L = 140, E = 50, and $F = \frac{1}{80}$.

$$\text{Then } G = \frac{140 + 50 + \frac{140 + 50}{5}}{140 - 50} \times \frac{1}{80} \times \frac{6}{5} = \frac{1}{26 \cdot 3}$$

or the gradient must be not less than 1 in 26.

In the above rule $\frac{L + E}{5}$ is added to allow for the friction of the drum and ropes; the amount of this, of course, varies with every incline according to the length, and character of the machinery, and K is added in order to give the preponderating weight necessary to give the required velocity.

This formula is for a simple jig, the full waggons running down on one line, and the empties running up on another. But on many inclines there is only one waggon-way; the empties are drawn up by a balance-weight of cast iron, which runs on a narrow way beside the waggon-way, or in the middle underneath the waggons (see Fig. 469).

This plan is frequently adopted for steep inclines, and is very convenient where there are a number of branch levels, as the waggon can be stopped at any level just as required, the empty taken off, and the loaded waggon hung on. The formula for a balance-jig differs from the preceding, and is as follows:—

Let B = balance-weight, and the other letters the same as before.

$L = 15$, $E = 6$, and $F = \frac{1}{110}$.
$B = 11$, $K = \frac{6}{5}$.

$$\text{Then } G = \frac{L + B + \frac{L + B}{5}}{L - B} \times F \times K = \frac{1}{11 \cdot 7}$$

In the above case the friction is given as $\frac{1}{110}$, because the waggon is carried on a trolley with large wheels. The accuracy of the foregoing rules is proved as follows: For the horse-road, the strain is to be equal going both ways.

Let T = the tractive force required for full road.
T′ = the tractive force required for empty road.

$$\text{Then } T = L \times F - L \times G$$

This is evident, because the tractive force equals the load × the fraction representing friction, and subtracted from this is the load multiplied by the fraction representing the gradient, because gravity is helping the train on.

$$T' = (E \times F) + (E \times G)$$

because the force required to draw the empties equals, of course, their weight × the fraction representing friction, and added to this their weight × the fraction representing gradient, because they are going uphill. But by the hypothesis $T = T'$;

$$\therefore L \times F - L \times G = E \times F + E \times G$$

and by transposition $(L - E)F = (E + L)G$

$$\text{and by transposition } G = \frac{(L - E)F}{E + L}$$

which is the rule.

In the case of the simple jig, leaving out K, the strain of the descending train is equal to the strain of the ascending train upon the rope.

Let T = the strain of the descending train.

T′ = the resistance of the ascending train, and the friction of both trains and machinery.

T = L × G; that is to say, it equals the weight of the loaded train × the fraction representing gradient.

$T' = E \times G + \left(L + E + \frac{L + E}{5}\right)F$; that is to say, it equals the weight of the empty train multiplied by the fraction representing the gradient, added to the weight of both full and empty trains multiplied by the fraction representing friction + $\frac{1}{5}$ of the weight of full and empty trains, added on account of the weight of the drum and rope multiplied by the fraction representing friction.

Since by the hypothesis T = T′—

$$L \times G = E \times G + \left(L + E + \frac{L + E}{5}\right)F$$

$$\text{or by transposition } (L - E)G = \left(L + E + \frac{L + E}{5}\right)F$$

$$\text{or by transposition } G = \frac{L + E + \frac{L + E}{5}}{L - E} F$$

which is the rule.

The result is multiplied by K in order to give the extra steepness required to obtain velocity. The rule for a balance-jig may be proved in a similar manner.

Fig. 445[1] shows plan and section of an ordinary jig. There are three rails from the top to the middle of the pass-by, then a double road; below the pass-by a single line, and at the bottom a double road; at the top a bank-head is made, that is to say, the roof is cut down to make a nearly level place. At the bottom a flat is made by taking up the floor at the start, the loaded waggons are pushed along over the hill where it is very steep, this gives them greater power to acquire velocity. Fig. 446 shows a jig drum, and Fig. 447 a jig wheel. It is perhaps hardly

[1] The plan and section are broken in the centre in order to shorten the drawing.

necessary to remark that the lightest gradient on which a jig can act depends partly on the weight of the train and partly on the

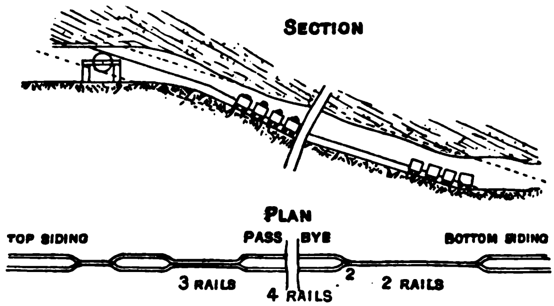


FIG. 445.—Jig: plan and section. Dotted lines show position of coal-seam.

length of the incline. The heavier the train the lighter the gradient on which it will be possible to run a train; the longer the incline

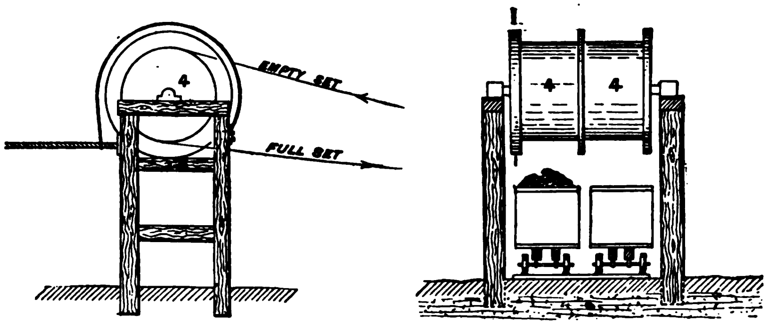


FIG. 446.—Jig-drums.

the heavier the ropes and drums, and consequently the steeper the incline required to make it work, unless the endless rope or chain system is adopted, in which case the length of incline will not be very material, as it will be in exact proportion to the weight of the train.

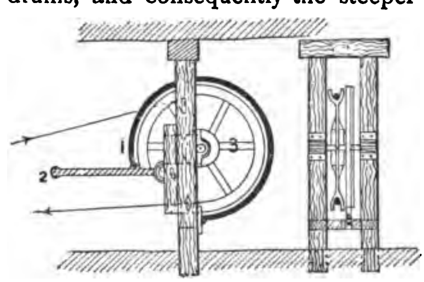
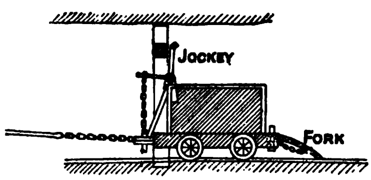


FIG. 447.—Jig-wheel. 1, brake-strap; 2, brake-lever; 3, jig-wheel.

Whilst the self-acting incline is the cheapest mode of haulage where the gradient is suitable, and horse-haulage is the cheapest for levels where the tonnage is not very great, engine-haulage must be adopted on all levels having a large tonnage, and upon all inclines where the load is drawn uphill. It has been said by mining engineers of great experience that it pays to use a steam-engine for haulage wherever the work on one road is such as to require five horses and five drivers. But this statement, of course, requires some qualification.

The following are some methods of hauling by engine-power.

Single Rope.—Where the incline is sufficiently steep for the waggons to run down, pulling after them a rope which has been coiled on the drum, a single rope and road are conveniently used; the waggons are lowered down to the bottom of the drift by means of a brake on the drum. For hauling up, the drum is connected to an engine by means of a clutch, which then winds up the drum. Any number of intermediate landings can be worked by this system by having points to guide the descending train into the landing, the engine-man being duly signalled which landing he is to stop at. Electric signals are generally used on long inclines as being easier and quicker to use than lever signals. Sometimes it is desired that the full train, after being drawn to the top of the incline, shall run without stopping to the pit-bottom. This may be effected by an automatic means of disconnecting the rope from the train at the top of the incline, as shown in Fig. 448.



Enlarged view of jockey and fork.

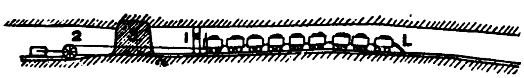


FIG. 448.—Single-rope haulage. 1, kicker-beam and jockey at top of incline; 2, single-rope drum and engine; 3, road to pit-bottom; L, fork.

A bell crank-lever, called a jockey, is clipped on to the front of the first waggon. At the top of the incline this strikes against a cross-bar, and, being thrown back, the lower arm of the lever pulls out a pin by which the rope is connected to the waggon.

Tail-rope Haulage (see Fig. 449).—Where the incline is not sufficiently steep or not sufficiently uniform for the empty train to run in-by all the way, it is necessary to pull it by means of another rope. This rope is generally attached to an engine near the shaft or top of the incline, but it is taken on pulleys

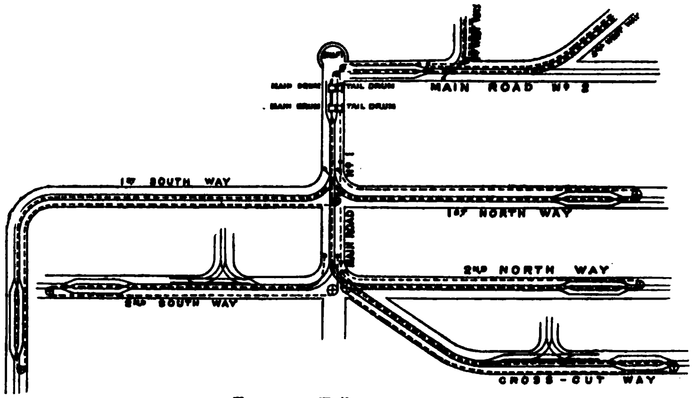


FIG. 449.—Tail-rope system.

fixed on one side of the road down to the in-by end of the engine-plane, and there passes round a pulley and comes back along the waggon-road, and is fastened to the front end of the empty set near the pit-bottom. This rope is called the tail rope, and is thus seen to be twice the length of the engine-plane. This tail rope is wound up on a drum by an engine, which in so doing pulls the empty train in-by. The main rope is hung to the rear of this train, and when it has got to the end of the plane the main rope is then unhooked from the empty set and fastened to the front of the full set, the tail rope being also unhooked from the empty set and hooked to the rear of the loaded set. The engine-man then winds up the main-rope drum, and the loaded set is drawn to the pit-bottom, pulling after it the tail rope.

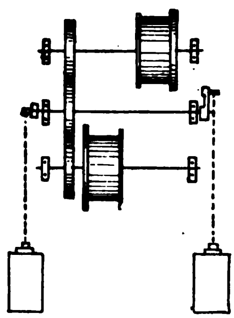


FIG. 449*a*.—Tail-rope engine.

Sometimes the main and tail rope drums (see Fig. 449*a*) are on separate shafts, connected by spur-gearing to the engine. By means of levers the spur-wheel on the engine-shaft can be thrown into gear with either the main or tail rope drum as required. In other cases both drums are loose on one shaft, and alternately fixed by clutches. Branch roads can be worked off the main road without difficulty. When the empty set is going in-by and gets to the branch, the tail rope is then disconnected, and another tail rope belonging to the branch is hooked on to the front of the empty set; the other end of the branch rope, which just goes as far as the beginning of the branch road, is then connected to the tail rope, the in-by part of which belonging to the main road is at the same time disconnected. The engine-man now continues to wind up the tail-rope drum, and the empty set is pulled forward by the branch tail rope, and therefore goes along the branch, the points being set accordingly. Any number of branches that is convenient may be used on this system.

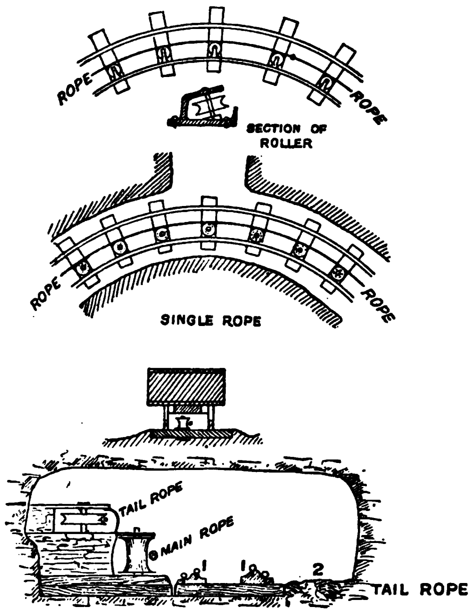


FIG. 450.—Methods of taking ropes round a curve. 1, check-rails; 2, travelling-road.

This method of haulage has been carried to great perfection. The underground railroads are laid with rails of great weight, from 25 to 50 lbs. to the yard, with fished joints and uniform gauge. At the curves there are check-rails and rollers (see Fig. 450) to guide the main rope round the curve as well as the tail rope. The speed of hauling often reaches 10 miles an hour.

Endless Chain.—In the endless-chain system there is a double waggon-way throughout. The empty waggons are always going in, and the full waggons always coming out. They are spaced in distances of from 10 to 30 yards apart, according to circumstances, but always equidistant (see Fig. 451). A chain is laid over the top of the waggons along the whole length of each road. The chain passes round a return wheel at the in-by end of the road, and round a drum near the pit. The drum is on a vertical shaft (see Fig. 451*a*), the chain passes four or five times round it, so as to give sufficient adhesion. The drum is like a pulley with a very wide and shallow groove. Across the groove are placed steel bars tapering inwards towards the bottom; a flange at the bottom prevents the chain from slipping off. The chain

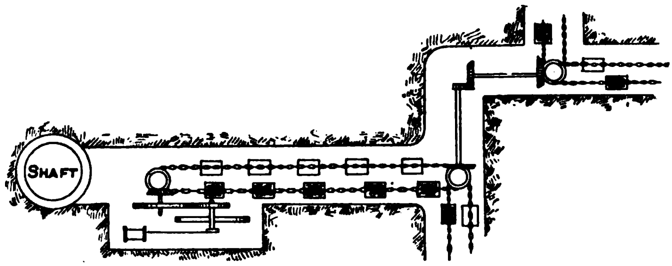


FIG. 451.—Plan of endless chain.

comes on at the top and off at the bottom, slipping down on these steel ribs; whilst one link lies flat upon the steel rib, the next link catches against it, and as there are a great number of these links catching against the ribs, the chain is prevented from slipping round. The weight of the chain lying upon the top of the boxes is sufficient to drag them along in case the road is level. But in many cases an iron fork is fixed at the back of the waggon (see Fig. 452), into which the chain drops, and through which the next link will not pass. With this kind of fork the waggons can be drawn up a very moderate gradient, of say 2 or 3 inches in the yard. If the gradient is much steeper, a short

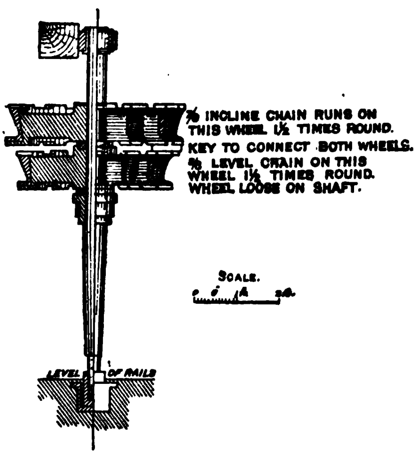


FIG. 451a.—Endless chain driving-wheels.

chain is used to attach each tub to the main chain; one end is hooked into a link in the large chain, and the other end is hooked to the waggon. As the chain is continually moving, and two roads are at work, a comparatively small speed suffices to convey a large tonnage to the pit-bottom. A speed of 4 miles an hour by this system will convey as much coal as a speed of 10 miles an hour with the tail-rope system. A common speed is about 2 miles an hour. The nearer the waggons are together, the greater the tonnage delivered for any given speed.

At the end of the main road there may be one or more branches or extensions. If that is so, the main chain is taken three or four times round a drum on a vertical shaft, and branch chains are taken from other drums on the same shaft, and these branch roads may again be extended in a similar fashion. As there is not so much coal coming down the branch road as down the main road, the tubs will be spaced at greater distances apart on the branch roads, and the driving-drum for these roads will also be of smaller diameter than the drum driven by the main chain. Where branches worked by horse enter the main road, the main chain may be lifted up and carried on pulleys, and flat sheets laid down on the main road, on which to turn the waggons coming out of the branch, and empties on the main road required

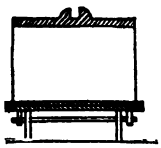


FIG. 452.—Fork on tub.

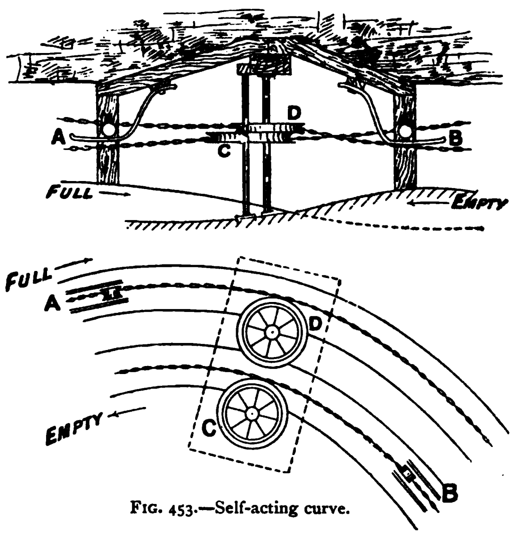


FIG. 453.—Self-acting curve.

for the branch. If there is a curve on the road, the chain is guided round by pulleys, but the waggons are detached from the chain whilst passing round the curve. The curve may be made self-acting, or nearly so, in the manner shown in Fig. 453, by slightly raising the full road as it approaches the curve; the waggons, having been detached from the chain, will run down again and reunite with the chain beyond the curve. The empty road is arranged in the same way.

The chain is sometimes placed in a hook on one side of the tub instead of passing over the top. This is more convenient where the coal is loaded above the top of the tub. In some cases, though rarely, the chain is placed on the ground underneath the waggon, being supported by rollers at short intervals. An example of this is given in the Report of the North of England Underground Haulage Committee. In this case the chain is working on an incline having a gradient of 18 inches in the yard, 600 yards in length. The waggons weigh 10 cwt. each when

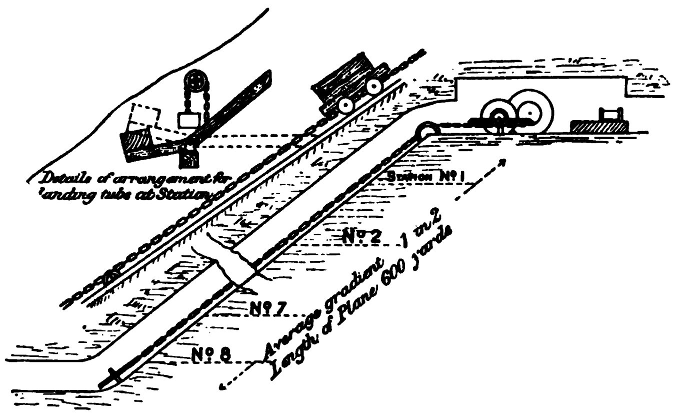


FIG. 454.—Endless chain: steep incline (South Wales).

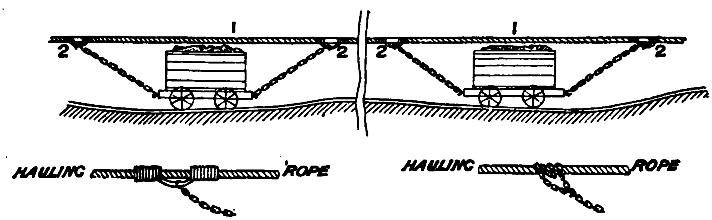
empty, and contain 15 cwt. of coal, placed 50 yards apart. The hauling-chain is made of iron 1¼ inch in diameter, and each link is 7 inches long; total weight, 22 tons. The drum, or driving-wheel, is 10 feet 3 inches in diameter. The engine has one cylinder 25 inches in diameter, 4 feet 5 inches stroke; the speed is reduced by gearing, so that the chain moves 2 miles an hour. There are eight stations on the chain. The waggon is attached by a chain 2 feet 6 inches long, of ⅞-inch iron, with a hook on each end. The landings are made by means of movable platforms, which can be lifted up by the aid of a balance-weight to let the waggons pass under, or drawn down to form a landing (Fig. 454). The empty tub running on to these is instantly unhooked and run out of the way. The full tub is run on to these and turned round, and hooked on to the main chain and drawn up the hill, the platform being raised again to let the next waggon pass under.

The endless-chain system is largely adopted, and is very economical; a comparatively inexpensive road will do, because of the slow speed of haulage, and almost any tonnage can be brought along a double road on this system. A comparatively small engine also is required, because the engine only has to drag the nett load of coals uphill. Of course, there is the friction of the whole system to overcome, but upon inclines the empty waggons going downhill balance the waggons coming up, all except the coals; and if the gradient should be uneven, the attachment of all the waggons on the haulage road to one chain equalizes the gradient, so that the engine has to pull the nett load of coals up an average gradient. If one of the inclines comes from rise workings, so that the load is going downhill and is connected to the chain system, the coals coming down this incline may balance other coals coming up another incline, and thus the power required for the whole pit may be no more than if it was quite level.

The objections to this system are that it is necessary to have a wide road for the double waggon-way, and wider still for men and horses to pass, unless there is a separate travelling road for them.

Endless Rope.—The principle of this is exactly similar to the endless chain, the difference is only in the detail. If the endless rope goes over the top of the tub, the tubs may be attached to it by short lashing chains, one end of which is hooked to the tub, and the other is twisted round the hauling rope (see Fig. 455).

Instead of tying this chain to the hauling-rope, a clip is often



Enlarged view of connection.

FIG. 455.—Endless rope: attachments.

used, of which Fig. 456 shows one example out of a great many. Another method is to fix an iron pin with a hook on the top of it into a socket on the back of the tub (see Fig. 457). The rope lies in the hook, and is thus carried over the top of the coals in the waggon. The strain of the hauling-rope on the hook causes

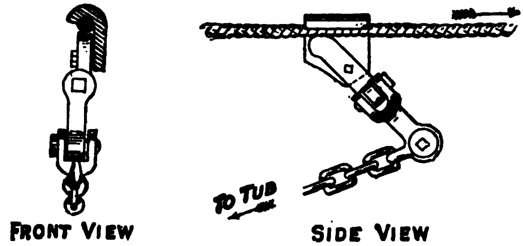


FIG. 456.—Endless rope: clip.

the pin to turn in the socket. This causes the sides of the hook to jam against the rope, which is so prevented from slipping. In

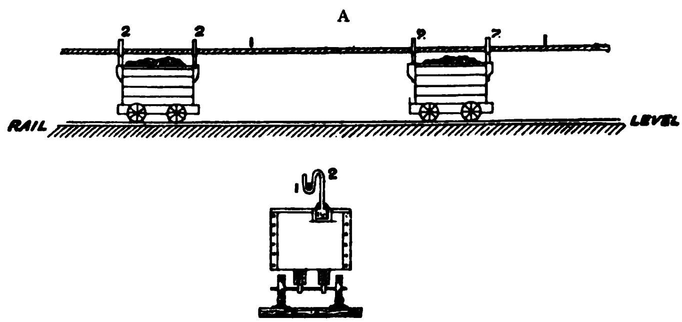


FIG. 457.—Connection of rope with tubs. A, longitudinal section; B, cross-section, showing connection. 1, hauling-rope; 2, connection between tub and rope.

many cases the rope is underneath the waggons, carried on rollers laid in the centre of the road; the waggons are then attached singly or in trains (see Fig. 457*a*).

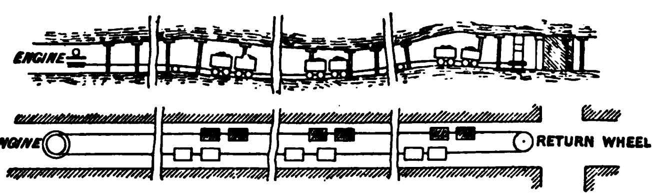


FIG. 457*a*.—Endless rope: haulage, general view: rope under tubs.

In the case of a long train, the rider holds a pair of tongs (see Fig. 458) fastened by a short chain to the front waggon. With these tongs he clips the rope, which then drags the train. To stop the train he opens the tongs. Sometimes the rope is clipped by means of plates brought together with a screw, and the train is sent on without a rider.

Another kind of clipping is made dependent on the resistance of the train, similar to that shown in Fig. 455; or in another way in Fig. 459. There are a great variety of these clips, and some of them are patented.

Movement is given to the rope sometimes by means of a

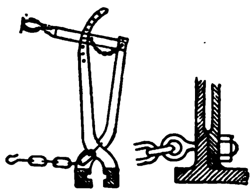


FIG. 458.—Endless rope: tongs.

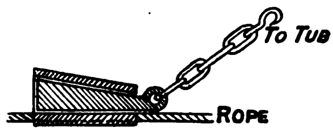


FIG. 459.—Ditto: wedge-clips.

drum known as Fowler's clip (see Fig. 460). In this case the circumference of the wheel is surrounded with a number of hinged clips which form the groove in which the rope rests. The pressure of the rope in the bottom of this groove by twisting the clips reduces the width of the groove, and so clips the rope tighter. The original width of the groove must be adjusted with some care to the size of the rope. In order that this may be

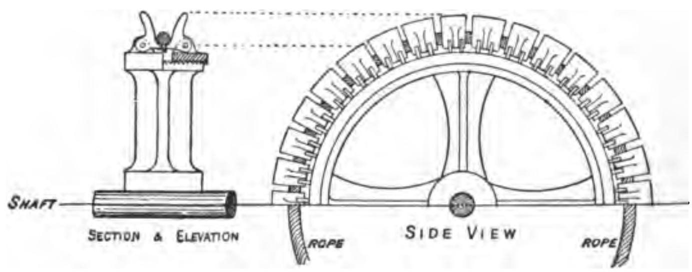


FIG. 460.—Fowler's clip pulley.

done, one row of clips is carried on a rim screwed on to the body of the pulley; by turning this screw the clips can be brought nearer together or further apart as required. The successful use of the pulley depends upon this adjustment. Another device is Barraclough's (see Fig. 461). In this case the periphery of the wheel is surrounded with short movable clips, but these slide on an incline instead of being hinged. They are brought together by the pressure of the rope, so giving the requisite adhesion; and sent out again by a spring under each pair of clips, as shown in section in the figure.

Another method, shown in Fig. 462, is to make the periphery of the drum or pulley wider than necessary for the rope, and then in this rim to put a crooked groove. The rope lying in this groove, bent by its twists, cannot easily be drawn through, but has to go round with the pulley or drum. Another plan[1] is to have an ordinary grooved pulley, and at intervals of say 18 inches a fork made of square steel about $\frac{5}{8}$ inch is placed in the groove, the shank passing through the body of the pulley, and secured by a nut on the under side. The edges of this fork catch against the rope and prevent it from slipping. This plan is said to be satisfactory by those who have tried it.

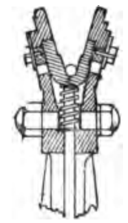


FIG. 461.—Barraclough's clip.

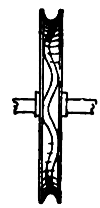


FIG. 462.—Grip pulley.

Another method is to use a very wide grooved pulley or drum, the lower flange of which is very deep. In this case the rope may be coiled five or six times in a spiral. As the drum turns, the rope comes off at the lower part of the groove, and comes on at the upper part. The tendency of the coil is to rise up, but it slips down the side of the groove. There is thus a continual slipping, but the ropes are found to wear remarkably well. Five or six years of hard work are recorded as the life of the rope working over these drums.

Another method of driving the rope is by means of a driving-pulley or drum and guide-wheel (see Fig. 463). Each of these

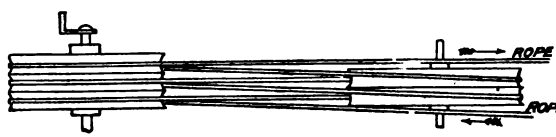


FIG. 463.—Endless rope: three-groove and two-groove wheels.

wheels is grooved to fit the rope, the driving-wheel having an extra groove, sometimes as many as six grooves in the driving-wheel to five in the guide-wheel. In working, the rope comes on to the driving-wheel, passing half round, and then over the guide-wheel half a turn, then in the next groove of the driving-

[1] Blackburn's patent.

wheel half a turn, again into the second groove on the guide-wheel, and so on—sometimes going straight from the driving-wheel to the guide-wheel, sometimes diagonally, forming the rope into the figure 8; but this figure-of-8 plan is more injurious to the rope. By a careful adjustment of the inclination of the guiding-wheel, the rope may leave and enter each groove of the driving and guiding wheels without any slipping on the flanges. All these methods of giving adhesion to the driving-rope are also adopted for self-acting inclines.

Some method of tightening the endless rope or taking up slack caused by stretching is usually adopted. The best plan is to take up the slack from the rope as it leaves the driving-wheel, because at this point there is no strain on the rope, which may be passed round a pulley fixed on a movable frame (see Fig. 464).

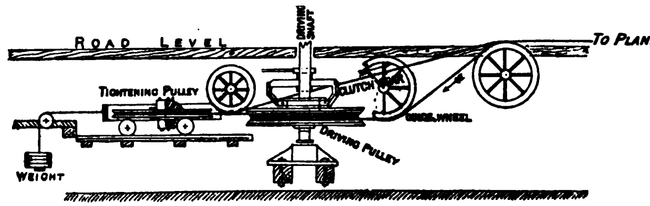


FIG. 464.—Endless rope: tightening wheel.

To this frame is attached a chain which passes over a pulley, a weight being hung at the end. This weight pulls the carriage and the tightening-pulley, and if an undue strain occurs the tightening-pulley gives way, and when the strain is less the slack is taken up instantly. The tightening-pulley, however, is often placed at the further end of the road in-by. In this case a heavier weight is required for tightening. It is inadvisable to have the rope tighter than is absolutely necessary, as this causes undue friction round curves and elsewhere. In some cases, with the rope working on the top of the tubs, it may be quite slack with advantage.

Application of Various Systems.—Each of the above systems of haulage has to be used or discarded according to the special circumstances of the case. Fig. 465 shows a plan of a mine in which various methods of haulage are adopted, each system being specially suited to the requirements of the district to which it is applied. It will be noted that there are six methods of haulage. The seam being thin, and the gate roads consequently low, hurriers are employed to bring the coal to the cross-level. Sufficient height is here made for ponies, by which the coal is taken to the engine or gravitation systems of haulage. The inclination of the seam being about 1 in 12, the coal is lowered from the upper cross-roads to the main level by self-acting inclines. The main level, having to carry a large tonnage, is worked by endless rope; No. 2 incline is worked by a single rope; No. 1

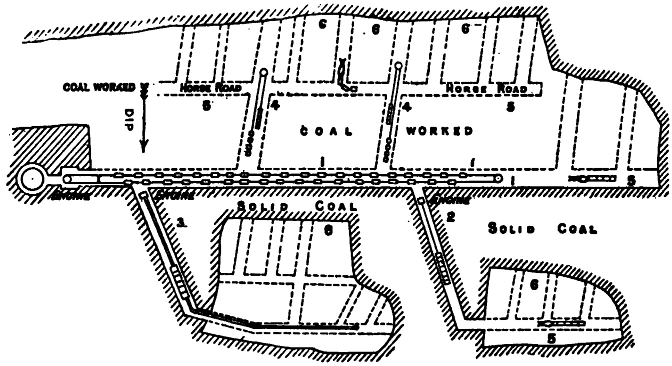


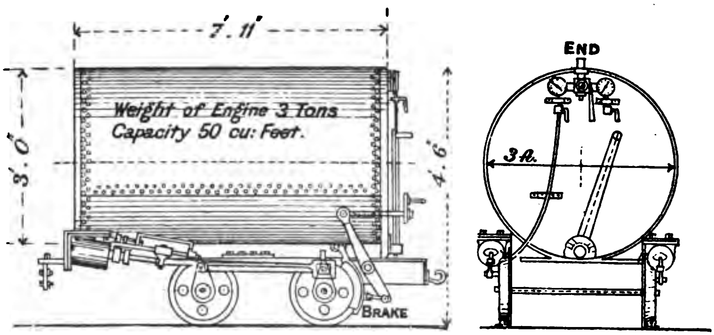
FIG. 465.—Plan of mine, showing methods of haulage. 1, endless-rope plane; 2, single-rope plane; 3, main and tail rope plane; 4, self-acting incline; 5, horses; 6, hurriers.

incline has a tail rope, because the lower level has been driven out for a length of over 1000 yards, and it is therefore convenient to apply engine-power for this length of level road.

Locomotives.—These are sometimes used in mines. The ordinary steam locomotive with the funnel cut down is sometimes used in American mines; it is, however, a very disagreeable and dangerous thing to have in the mine. When it gets off the rails there is danger of setting fire to the timber, and in any case it fills the road in which it works with smoke. This road must, of course, be the return-air road, as, if it were in the intake road, the mine would be unbearable.

Compressed-air Locomotives.—To overcome the annoyance from the use of steam locomotives, compressed air is used. Such an engine is similar to a steam locomotive, but there is no fire, and the boiler is a simple shell or cylinder of steel, which is filled with air at a high pressure, say from 200 to 400 lbs. on the square inch. These have been largely used in the long railway tunnels through the Alps. A smaller kind was patented by Messrs. Lishman and Young, of Durham, and was set to work at the Newbottle Colliery.

These machines were of three sizes, the smallest with cylinders 3 inches diameter, weighing 17 cwt.; the next, cylinders $3\frac{1}{4}$ inches diameter, 6 inches stroke, receiver holding 40 cubic feet, and the total weight being 27 cwt.; and the largest, cylinders $4\frac{1}{2}$ inches diameter, 8 inches stroke, receiver holding 50 cubic feet, total weight 3 tons (Figs. 466, 467). The pressure of air used was about 200 lbs. Large air-compressors on the surface force the air into pipes going down the shaft and passing along the level worked by the locomotives. At the pit-bottom, and at several stations along the level, were fixed branches and taps, at which the locomotives could take in a fresh charge by means of a union-screw coupling; by this means a fresh charge is taken, in from about fifteen seconds to a minute, the latter time being



FIGS. 466, 467.—Compressed-air locomotive.

required to completely fill the receiver. But by allowing a difference between the pressure in the locomotive and that in the pipe, the time occupied in charging is greatly reduced. The locomotives will run a distance varying from a quarter to half a mile, or more if necessary, without taking in a fresh charge. The advantage of the locomotive is that it will go along very crooked roads as long as the rails are carefully laid at proper curves, and that only a single road is necessary for a considerable traffic.

The disadvantages are the necessity of maintaining a first-class road to avoid derailment, and that the system is not adapted to any but level roads. It is well known that the locomotive works at a great disadvantage on an incline even as moderate as 1 in 30. Therefore this mode of conveyance should only be introduced in those places where the gradient of the intended haulage road and its possible extensions is known to be level.

Electric Locomotives.—As long ago as 1883 the writer saw an electric locomotive working at the Zaukeroda Colliery, near Dresden. Since then one or two more electric locomotives have been set to work in European mines, and some have also been adopted in the United States.

Fig. 468 shows the arrangement at Zaukeroda. On the surface is a steam-engine with cylinder 10 inches diameter, 8 inches stroke; the fly-wheel makes 200 revolutions a minute, and by means of a belt the dynamo is driven 660 revolutions. Two copper conductors pass down the shaft 242 yards deep, and are there connected with iron conductors fixed to the timbers in the roof of the level, which is 792 yards long; one of these conductors is for the positive or intake current, and the other for the negative or return current. The locomotive contains a motor connected to the axle by spur-gearing. The current is taken from the

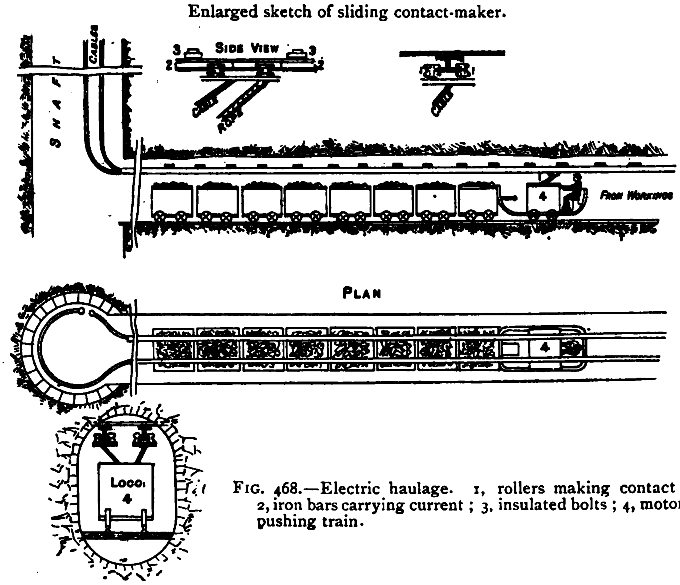


FIG. 468.—Electric haulage. 1, rollers making contact; 2, iron bars carrying current; 3, insulated bolts; 4, motor pushing train.

iron conductors to the locomotive by means of brass travellers, which grip the lower end of the iron conductor with four rollers; a copper wire passes from the traveller to the motor. As the locomotive moves it drags the traveller along. In order to save the copper wire from undue strain, a short rope is fastened to the travellers and the locomotive. The speed of the loaded train was 6 miles an hour, and that of the empty train 8 miles. Each train consisted of eighteen waggons weighing each when empty 5 cwt., and holding 10 cwt. of coal.

In more recent electric locomotive haulages, some modification has been made in the conductors and mode of connection with the locomotive, but the chief features remain the same.

On every incline provision must be made against accidents through waggons becoming uncoupled or the rope breaking. The precautions taken to effect this are as follows :—

Behind the train is hinged a strong iron fork, which is dragged along by the train. The construction of this fork is such that if the rope should break, and the train should back, the points of the fork will stick into the ground, and lift up the lower end of the waggon and throw it off the rails, thus bringing the train to a stop before it has acquired any great velocity.

Another device is to have points and crossings so arranged that the descending train will be thrown off the road. This, again, is only suitable for a road where the traffic goes only one way. But points may be laid in a single road with traffic going both ways, that can be opened when required to throw the train off the road. The point is moved by a lever connected by a wire cord with the top or bottom of the incline, so that the brakesman or hanger-on can open these points if there is a runaway train. At the top of each incline are movable stop-blocks, which must be always placed in position when the train is at the top.

Where a long train of waggons is coupled together, a safety-chain is often laid over the tubs from end to end, and fastened to the rope in front and to the rear of the waggon, so as to prevent any of the tubs from running back in case a draw-bar or coupling should break.

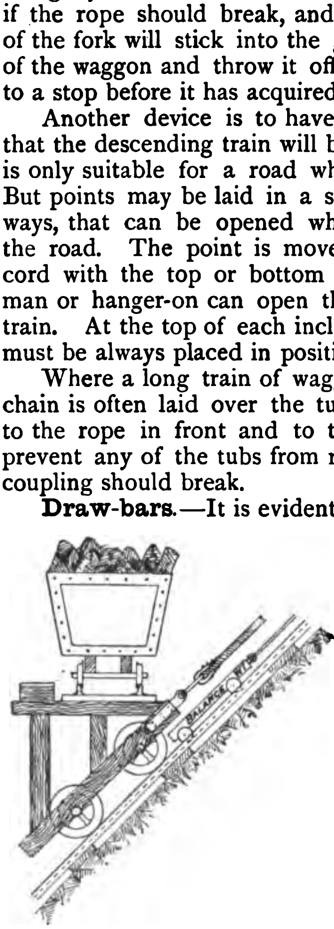


FIG. 469.—Slope-carriage and balance-weight.

Draw-bars.—It is evident that the draw-bar of a pit-waggon must be made to suit the method of haulage. Where, for instance, the endless-chain haulage is adopted throughout, and the waggons are brought from the workings on to the chain by boys, no draw-bar at all is required. But if half a dozen or more waggons are brought to the chain by means of a pony, then the draw-bar must be strong enough to stand such a strain as the pony can apply. But if the waggons are drawn up an incline by an engine, or lowered down by a brake in long trains, then the draw-bars must all be strengthened, so that when the rope is attached to the draw-bar of the front waggon, and has to pull a train of perhaps forty or fifty or even a hundred waggons behind it up a steep incline, it may be strong enough to perform this work. Thus it is sometimes necessary to renew all the draw-bars at a colliery when a new system of haulage is introduced.

Slope Carriages.—Where the incline is very steep, it is not only very difficult to get the waggons on to it from a level road, but there is a tendency for the mineral to fall out. To overcome this difficulty a slope-carriage (Fig. 469) is used. This has a

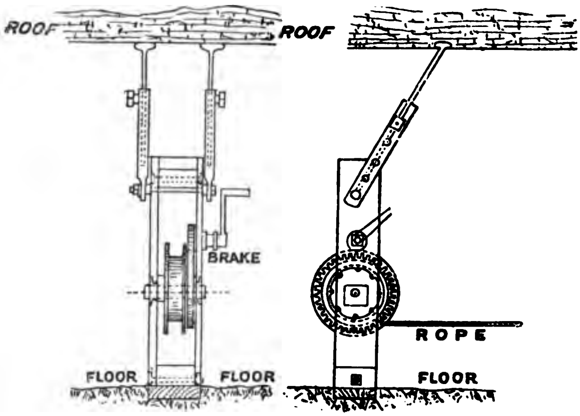


FIG. 470.—Jack roll or winch.

level top, and is brought to rest opposite the landing. If there are intermediate landings between the top and bottom of the incline, the waggon is run on *across* the slope-carriage, as shown in the figure; if there are no landings between the top and bottom of the incline, the waggon may be run on or off any way.

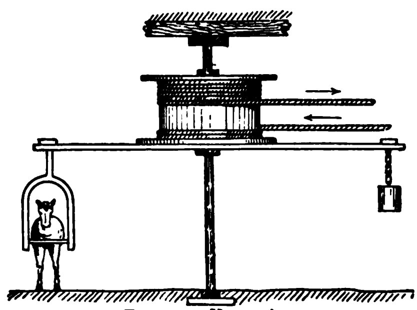


FIG. 471.—Horse-gin.

For a short incline close to the working place small winches are sometimes used, worked by hand, and spragged between the roof and floor, as shown in Fig. 470, a thin steel rope being used. Where the mineral has to be hauled up an incline where steam-power is not available, a horse-gin is sometimes used, as shown in Fig. 471. Hauling-engines are often very conveniently and cheaply fixed on timbers, as shown in the small sketch (Fig. 472). Fig. 473 shows some rope rollers which can be fixed to timber props or bars, to carry tail ropes or endless ropes used for transmission of power; one of these is hinged, so that it may adjust itself to the line of the rope.

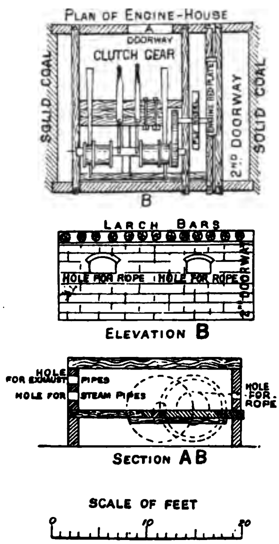


FIG. 472.—Hauling-engine on timbers.

Electric Motors.—Electricity is now used at several places for driving underground hauling-drums. Fig. 474 shows plan and elevation of a steam-engine and dynamo of 40 H.P.; several plants of this description are now in operation. The electricity passes through carefully insulated copper conductors made of nineteen copper wires of 15 Birmingham wire-gauge; they are in an insulated covering, making each cable 1 inch in diameter. From the motor to the pit-bottom they are in a wooden casing made of pitch pine, 6¾ inches by 3¼ inches external dimensions (see Fig. 476), made of two pieces, the first 6¾ inches by 2 inches, with two grooves 2½ inches apart, in which the cables are laid; over these a piece 6¾ inches by 1¼ inch is screwed with 3-inch brass screws. This casing is fastened to the shaft-side. Fig. 475 shows the electric motor and drums. The power is taken from the electric

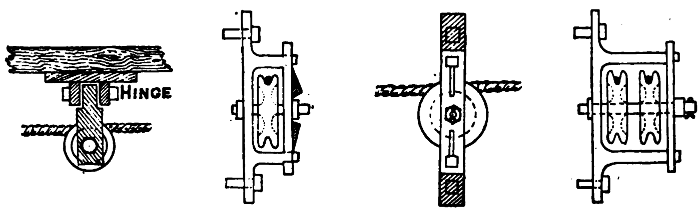


FIG. 473.—Rope rollers.

motor by means of ropes working in a grooved pulley on to a large grooved wheel on an intermediate shaft; a pinion on this intermediate shaft moves a spur-wheel on the drum-shaft. Smaller electric winches or hauling-engines from 5 to 15 H.P. are also made.

Pit-bottom.—The rapidity of haulage and winding depends a good deal on the arrangement of the railways at the pit-bottom. A suitable arrangement is shown in Fig. 477.

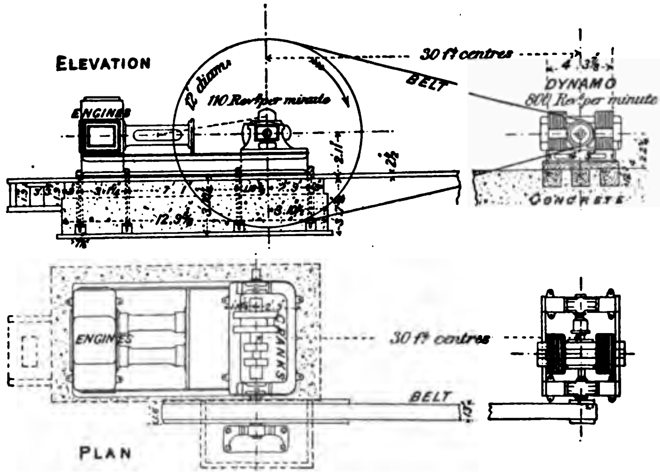


FIG. 474.—Electrical hauling-plant.

Ropeways.—For the conveyance of minerals on the surface, it is not always convenient to make a railway. A ropeway is sometimes made by which roads, rivers, and chasms can be bridged (see Fig. 478). Some details of the posts on which the buckets are carried are shown in Fig. 479, and on a larger scale in Fig. 480. This is the Otto system largely used in Germany. The ropeway consists of a bearing-rope stretched from post to post, one on each side forming a double road; rollers, from which the buckets are suspended, run over this bearing-rope; below this is an endless driving-rope resting on rollers on the bucket-frames, and clipped to these frames when the buckets are travel-

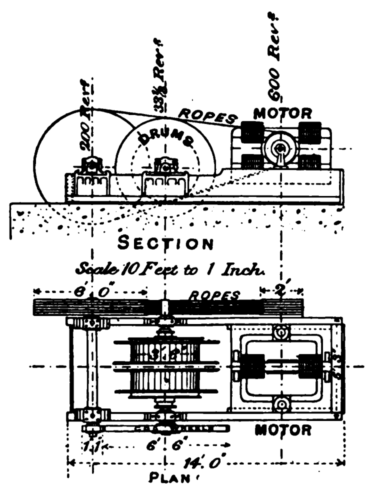


FIG. 475.—Motor and drum.

ling. When properly used this system gives great satisfaction. The plan has been adopted, not merely in mountainous regions, where the construction of an ordinary tramroad would involve great expense, but on the plains of Germany, where it has been applied to a large colliery to convey the mineral from the pit to the railway. Owing to the absence of all earthwork, bridges, level crossings, fences, or other interference with the use of the ground, except the planting of the necessary posts, this method of convey-

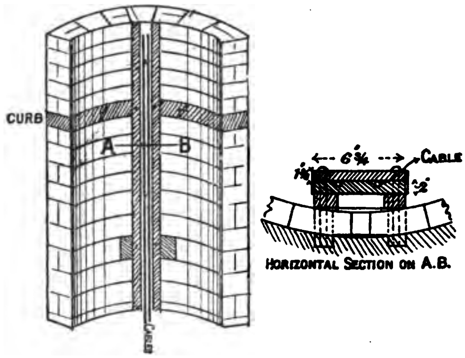


FIG. 476.—Electric cable: casing in shaft.

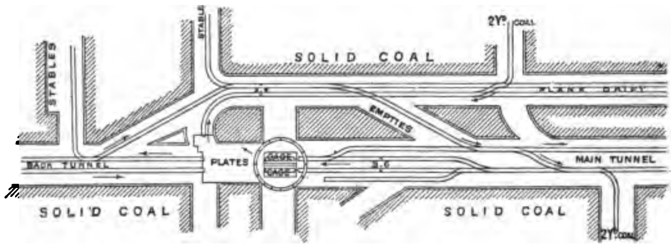
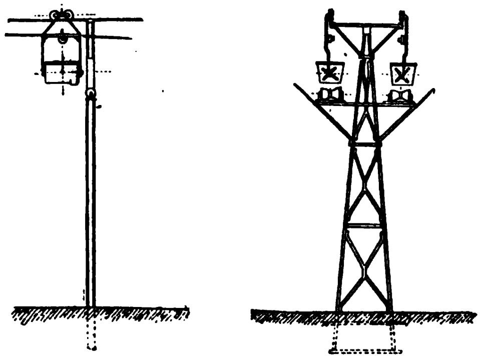


FIG 477.—Pit-bottom siding arrangements.

FIG. 478.—Ropeway: general view.

ance is, in many cases, the cheapest that can be adopted in first cost, and it is doubtless also the cheapest in working cost.

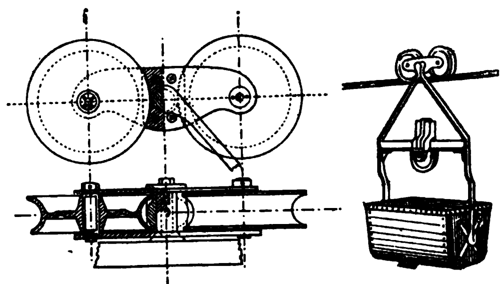
Another system of ropeway, which, however, is not so generally suitable, consists in having only one endless rope. The bearing-



Scale 20 Feet to 1 Inch.

FIG. 479.—Ropeway: posts.

rope, in this case, is an endless rope driven by an engine; the buckets are suspended by hooks to the rope, which is carried on rollers at the posts. There is, however, a liability of the buckets



Scale 1½ Feet to 1 Inch

FIG. 480.—Ropeway: rollers and buckets.

to slide along the rope when approaching the posts, owing to the steepness of the curve. This is especially the case where the gradient of the ropeway itself is steep.

CHAPTER XVII.

TRANSMISSION OF POWER: RODS, ROPES, AIR, WATER, ELECTRICITY, OIL.

POWER is transmitted from steam-boilers on the surface into the mine in the following ways:—

Steam.—This is taken down the shaft in pipes to work engines near the bottom, and is also conveyed along the roads to considerable distances, in some cases as far as 1200 yards. In order to prevent excessive condensation, the pipes must be thickly covered with non-conducting composition. Expansion joints must be introduced, and in the levels and inclines the pipes must be carried on rollers or suspended between the expansion joints; otherwise there will be continual breakages of joints. This is probably the cheapest mode of transmitting power into the mine, but there are several objections. A leakage of steam is annoying; should the steam-pipes burst, the steam might be dangerous, and the pipes must be so placed that in such a contingency the steam will not pass through any working place. The heat from the steam is often inconvenient, and in some cases would be apt to cause spontaneous combustion.

The percentage of loss from condensation depends on the amount of power transmitted, because the actual loss is proportional to the circumference of the pipe from which the heat is radiated. But the friction of a gas in a pipe varies inversely as the fourth or fifth[1] power of the diameter, so that whilst a 2-inch pipe might be sufficient for 10 H.P., a 4-inch pipe would be sufficient for 50 or 60 H.P. In the latter case the radiating surface of the iron pipe is double; the radiating surface of the covering composition is, however, only one and a half times, whilst the power transmitted is from five to six times as great. Therefore the loss taken in percentages will be about one-third for the larger power of what it was for the smaller power. If, in the first instance, 60 per cent. of the steam was condensed, in the second instance only 20 per cent. would be condensed.

[1] It is generally taken as the fifth power.

Professor Merivale has given a good deal of attention to this question, and the reader is referred to his interesting papers in the "Transactions of the North of England Institute of Mining Engineers," vol. 35, part 3, and vol. 36, part 1. Mr. Merivale describes an installation where the steam from a cylindrical boiler, 29 feet long and 5 feet in diameter, is taken down a shaft to drive three engines underground. The first engine, 12-inch cylinder, 2-feet stroke, is 342 yards from the boilers, and the steam passes through 5-inch pipes. The second engine is 950 yards further, supplied by 2½-inch pipes; the engine is 8-inch diameter, 1-foot stroke. The third engine, 120 yards further, 1½-inch steam-pipes, is 6-inch diameter, with a 10-inch stroke. The total distance is 1414 yards from the boilers. The pipes, steam-traps, and engines are covered with Wormald's composition, and in the shaft, which is rather wet, the pipes are covered with felt and lead. The engines in the pit work from 5 hours to 11 hours a day. The boilers consume 427 lbs. of rough small coal per hour, and evaporate 273 gallons of water at 56° at a pressure of 35 lbs. Of this 273 gallons, 57½, or about 21 per cent., are collected at the steam-traps in the pit, showing that the total loss by priming and condensation is only 21 per cent. of the steam-power. It is, however, probable that the loss by condensation is rather greater than that indicated by the water condensed in the steam-traps, as some of the condensed water may very likely pass through the steam-engines. However that may be, there is no doubt that this installation of Professor Merivale's demonstrates that steam may be carried with great economy to a distance of nearly a mile, and if, instead of a small horse-power and intermittent working, there had been a large horse-power and continuous working, the loss by condensation would have been very much less for the reasons given above.

Compressed Air.—Compressed air is the most usual and most generally approved method of transmitting power to mines, because it can be taken into any part of the mine without danger or inconvenience, and applied to any kind of work, whether pumping, hauling, winding, ventilating, coal-cutting, or rock-drilling (see Fig. 481). The air-compressor is simply an air-pump (see Fig. 482). The air is sucked in at one valve, and driven out from another valve into a receiver. 1, 1, are the inlet valves, and 2, 2, are the delivery valves. In case cooling water is admitted into the cylinder, the delivery valves must be at the bottom. An arrangement of engine and air-compressor erected by the writer is shown in Fig. 482*a*. In this case the steam-cylinders had a diameter of 24 inches, and the air-cylinders 25 inches, and the stroke 4 feet. They were coupled engines.

The air-pressure can be raised to a higher pressure than the steam if desired. In this case the boiler pressure was between 40 and 50 lbs., and the air was compressed to about the same pressure. In order to conduce to the economy of steam, a cut-off valve,

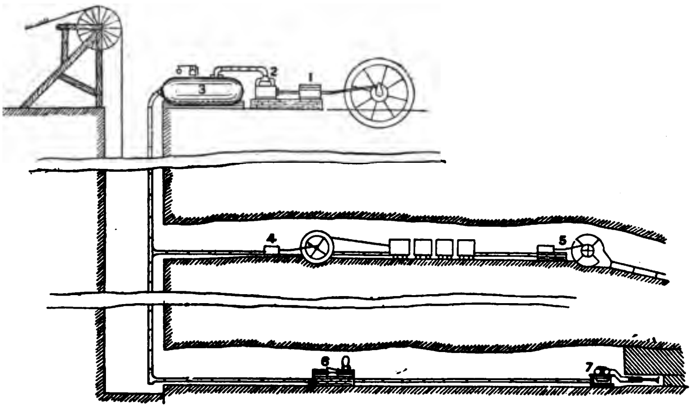


FIG. 481.—Transmission of power. 1, steam-engine; 2, compressor; 3, air-receiver; 4, hauling-engines; 5, fan; 6, pump; 7, coal-cutter.

worked by cam-gearing, was fixed on the steam-pipe. In order to maintain the air in the receiver at a uniform pressure without the constant attendance of an engine-man, the throttle-valve was controlled by a governor; the governor consisted of a piston working in a small cylinder on the air-receiver, which lifted a weighted lever and was connected with the throttle-valve. If the air-pressure fell, the throttle-valve was opened wider; if the air-pressure rose, the throttle-valve was closed, and in this manner the air-pressure was maintained nearly uniform. In case, however, an air-pipe should burst (a most unlikely contingency) and the air-pressure rapidly fall, the governor lever would automatically disconnect itself from the throttle-valve, which would be instantly closed by the action of a weighted lever. Where inlet valves, similar to those in the sketch, are used, there must be a guard of perforated plate inside the valves, so as to obviate the possibility of a broken valve falling into the cylinder.

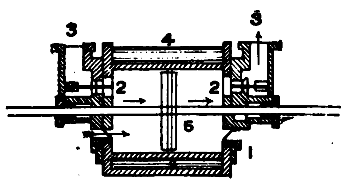


FIG. 482.—Transmission: air-cylinder. 1, inlet valves; 2, valves to receiver; 3, pipes to receiver; 4, water-jacket round cylinder; 5, air-piston.

The compression of air produces a great deal of heat, just as the expansion produces cold. In order to keep the cylinders cool, it is usual to place them in a tank of water, through which a stream is maintained. Sometimes a jet of water is admitted into the cylinder. This jet should be in the form of a very fine

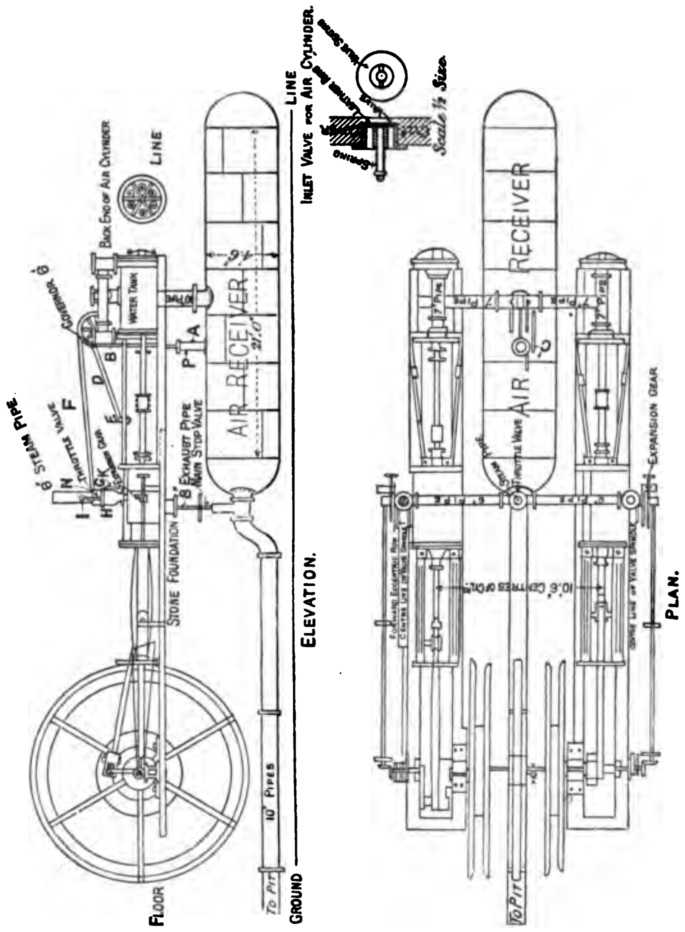


FIG. 482a.—Air-compressing engine.

spray, the finer the better, as it then mixes better with the air, and the cooling is more complete with a less quantity of water. A great quantity of water in the cylinder, as all mechanical engineers know, is a danger to the safe working at a high speed, and a great many engineers prefer to keep the cylinder dry. In some cases water has been introduced between the piston and the air (see Fig. 483). The air is here on the top of a column of water, forming a fluid piston; the piston raises the water and compresses the air above it. This design has not come into general use, because water is only suitable for a piston when moving at very low speeds.

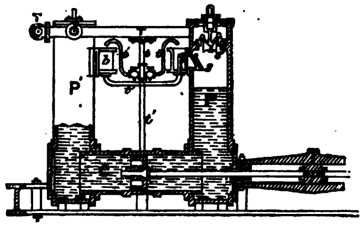


FIG. 483.—Transmission: water in cylinders.

The higher the speed at which an air-compressor works the better, because there is a less percentage of leakage through the valves and past the piston, and there is less time for the heating of the inlet air.

It is important that the clearance at each end of the cylinder should be as small as possible, in order that the compressed air may not be left in; because, on its expansion, it occupies a place which should be filled by fresh air, and so reduces the useful work done by the pump. If too little clearance is left, however, the piston may knock.

In order to get over the difficulty of clearance, the following ingenious device has been adopted by some engineers: At each end of the cylinder is a groove, rather longer than the thickness of the piston. When the piston has passed the end of this groove, the air on the compression side passes through this groove on to the intake side, thus compressing the air in the cylinder already full, and reducing the pressure on the side which is about to become the inlet side of the piston. The objection to this system is that it may cause a knock of the piston against the covers, unless there is sufficient cushioning in the steam-cylinder.

The economical use of compressed air depends on the following rules:—

The boilers and steam-engines used for the purpose on the surface must be of the best kind. It is vain to seek an economical system of transmitting power unless the power is economically produced in the first instance. It is quite common for colliery engines to consume two or three times as much fuel as is necessary; and as long as that is the case, it is not worth while to give too much attention to the relative economy, as distinguished from the safety and convenience of rival systems of transmission, where the superiority claimed over one system by the other is, perhaps, not more than 10 or 20 per cent. Having got economical steam-engines, the next step is to have the compressors with air-tight pistons and valves, taking care that the inlet and delivery valves are of sufficient size.

The inlet valve must be open to the atmosphere, so that the air may enter cold, and not be taken from a warm engine-room; and the delivery port must lead straight away from the cylinder, so as to keep the hot air away from unnecessary contact with the cylinder.

The heat produced by compression represents loss of power, and this loss is greater the higher the pressure. On this account, a low pressure, of say 30 lbs., is more economical than higher pressures; but this economy is counteracted by the necessity of having larger pipes and larger air-cylinders than if higher pressures were adopted. If the air is compressed to a higher pressure, say 60 or 80 lbs., and used expansively, considerable economy may be obtained. With the ordinary mining compressors, the indicated power of the air-engine in the mine is, perhaps, only 20 to 40 per cent. of the steam-engine working the compressor. But there is no reason why an efficiency of 50 per cent. should not be obtained with care.

The student may derive some information on the compression of air and the working of air-engines by the study of the diagram, Fig. 483*a*. This shows what takes place in the cylinder of an air-compressor, which, for the sake of convenience, is supposed to be 8 feet long. When the piston has gone half a stroke, the pressure will be doubled; when it has gone three quarters of a stroke, the pressure will be quadrupled, because the space the air has to occupy is one-fourth of the original space. As the piston gradually moves along, the rise of pressure is gradual. This is indicated by a curved line, which is called on the diagram the isothermal curve, letters *a*, *a*, *a*, *a*. There is another curve marked on the diagram, called the adiabatic curve, letters *b*, *b*, *b*, *b*. The isothermal curve shows the gradual rise of pressure with the movement of the piston, supposing that the heat of air is constant throughout the stroke. But if the heat, which is always produced by compressing air, cannot escape, it will cause the air to expand, and consequently the pressure to rise. This rise of pressure is shown by the adiabatic curve, *b*, *b*, *b*, *b*. If, after the stroke has been completed, the air is cooled, it will contract to the space within the isothermal curve. And the area enclosed between the isothermal curve and the adiabatic curve represents power that has been wasted during compression, due to the heating of the air (this heat being lost, in ordinary working, by cooling in the air-pipes). Supposing that the engine had stopped just as it completed five-eighths of the stroke, all the heat having been left in the air, the pressure in the cylinder would have been four atmospheres, or 60 lbs. If the engine were disconnected from the compressor, and the air in the compressor now allowed to expand, it would push the piston back to the starting-point (it is assumed, for the sake of argument, that the piston has no friction), the pressure would go down to atmospheric pressure, and the temperature to the original temperature, all the heat generated by compression being reabsorbed in the expansion of the air. If we now assume that the engine makes another stroke of the compressor, and that this time the air is cooled during compression, then the pressure will rise to four atmospheres when

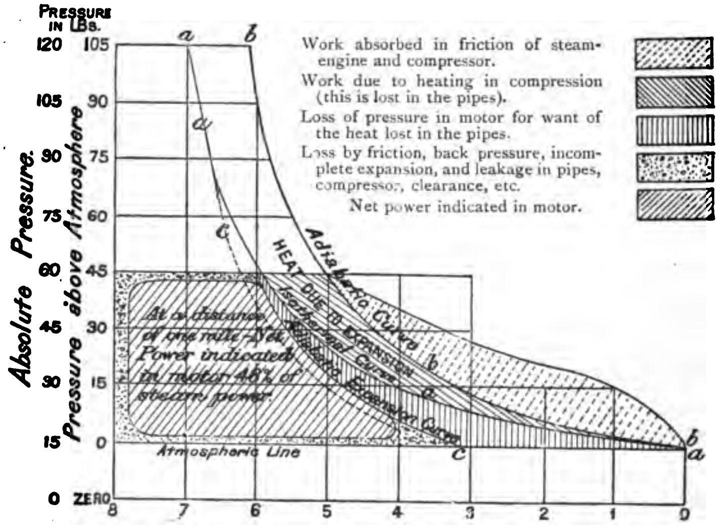


FIG. 483*a*.—Work done by steam in compressing air, shown by all the shades.

the piston has made six-eighths of the stroke. If the engine is again disconnected, and the air in the compressor allowed to expand, it will force the piston back, but not to the starting-point (assuming that no heat enters the cylinder), because the expansion of the air will cool it, and the cooling will contract it, so that the expansion of the air will take place as shown by the curve, *c*, *c*, *c*. And, in order to send back the piston to the starting-point, some heat will have to be put into the cylinder, equal to the amount that was taken out during the compression-stroke.

It is thus evident that the loss of power, due to the heating and subsequent loss of heat, in using compressed air is nearly double (say 1·76) the amount represented by the space between the adiabatic curve and the isothermal curve. The first loss is in compression, when the resistance to the engine is increased by the expansion due to heating; the second loss is in the air-motor, where the expansion of the air is prevented by lack of heat. This loss in compression may, as already explained, be to some extent reduced by a properly arranged water-spray. As generally made, these water-sprays have a very slight effect. It is stated, however, that a new water-spray has been invented which has a good effect. In this case, the water is forced in at a very high pressure, forming a very fine spray *during the compression.* The loss in expansion is also, to some extent, modified by the absorption of heat through the metal of the cylinder of the air-motor, and also by stoves heating the air, where the use of fire or open lights is admissible. Assuming, however, that the double loss of power is not mitigated in any way, the effect of first-class air-compressors and air-motors of not less than 100 H.P. of steam, with air at an absolute pressure of four atmospheres, or a working pressure of 45 lbs., will be about as follows:—

Generating engine	100	H.P.	
Friction of engines and compressors ...	20	,,	= 20 % of indicated power of engine.
Loss in heat	21	,,	= 25 % of power in air.
Loss in leakage of compressor-piston, valves, and in clearance	4	,,	= 5 % of air.
Friction in pipes, length say one mile of 6-inch pipes	2	,,	= 3 % of air.
Leakage in pipes	2	,,	= 3 % of air.
Loss by incomplete expansion of air in motor	3	,,	= 6 % of power in air at motor.
Friction of motor piston, bearings, gearing, etc.	8	,,	= 17 % of indicated power in motor.
Useful effect on hauling-drum or pump ...	40	,,	Thus 8 + 40 or 48 % of engine power is indicated in motor.
	100	,,	

A system of compound compressors is sometimes adopted; that is to say, the air is compressed in one cylinder, to say 30 lbs. pressure above the atmosphere, and is then conveyed in pipes to another cylinder, where it is compressed to say 120 lbs. But between the two cylinders the air is cooled to the atmospheric temperature. In a similar way, the air may be used expansively in compound air-engines. The high-pressure air is used in a small cylinder, and the exhaust air is conveyed through pipes to a larger cylinder; but on the way it is reheated by the atmosphere nearly to the átmospheric temperature, and thus takes back from the air some of the power lost in compression.

In the case of engines working on the surface, the air is heated, by passing through a small stove, to a temperature of about 300° Fahr. By this means great economy is obtained, because the cost of the fuel used in the stove is insignificant, and after allowing for the cost of this, the air-engine may indicate 70 per cent. of the power in the steam-engine. Where the use of such a stove can be permitted, there is little doubt that compressed air is the most economical method of transmitting power that is known. In coal-mines, as a general rule, the use of the stove is out of the question; but it is, perhaps, nearly as safe as an oil-engine or electric motor, and is safer than some of the methods now used to prevent the freezing of the ports.

A system of air-compression by water-power has recently been adopted in Canada, which promises better results that any other system yet adopted. In this case the air is introduced into a falling current of water, the height of the fall equalling the head of water necessary to give the required pressure of air. At the bottom of the fall the air escapes into an iron chamber, whence it is conveyed by a pipe up the shaft to the works. The water rises up another pipe, or up the shaft, to the outfall. The level of this outfall is of course lower than the inlet, and the available water-power in foot-pounds can be calculated by multiplying the weight of water in pounds by the difference of level in feet between the inlet and outlet of the stream of water. It is stated that 70 per cent. of the energy in the waterfall is represented in the compressed air.

The required head for compressing the air is got by sinking a pit. The required difference in level of the inflow and outflow water-currents, which may be only a very few feet when the volume of water is large, may be obtained either by a canal or tunnel, as in the case of a river, or, where the same water is used over again, as in the case of a cooling reservoir, by means of a pump driven by a steam-engine. The advantage of this method of compression is to be gained by the cooling of the air as it mixes with the falling water and the absence of the friction of pistons and valves.

Air-pipes, Friction.—It is important to know the quantity of compressed air required for any given horse-power, and that is shown in the following table :—

TABLE XIX.

Modes of applying the compressed air in consumers' engine.	Quantity of air at 45 lbs. pressure (that is, 4 atmospheres absolute) required per ind. H.P. per hour.
	Cubic feet.
CASE 1.—Where air at 45 lbs. pressure is reheated to 320° Fahr., and expanded to atmospheric pressure	125·4
CASE 2.—Where air at 45 lbs. pressure is heated by boiling water to 212° Fahr., and expanded to atmospheric pressure	145·4
CASE 3.—Where air is used expansively without reheating, whereby intensely cold air is exhausted, and may be used for ice-making, etc.	188·4
CASE 4.—Where air is heated to 212° Fahr., and the terminal pressure is 11·3 lbs. above that of the atmosphere	240·6
CASE 5.—Where the air is used without heating, and cut off at three-quarter stroke, as in ordinary slide-valve engines	258·0
CASE 6.—Where the air is used without reheating and without expansion	331·8

The next important question is the diameter of the pipe required for the transmission of a given volume of air. A good deal of attention has been given to this question, especially by continental engineers, and the general conclusion seems to be that the friction of air is governed by the same rules as those which govern the friction of water; and, in fact, that the friction of all fluids is governed by the following rules:—

The friction is in direct proportion to the density of the fluid. Thus, water being eight hundred times as heavy as air at atmospheric pressure, the friction at equal velocities of water will be eight hundred times greater than that of air. If, however, the air is compressed to eight atmospheres, thereby increasing its density eightfold, the friction of water will only be one hundred times as great as that of the compressed air at equal velocities. Following the same rule, it is found that the friction of steam is less than that of air, because it is lighter, the density of steam at equal pressures being approximately half that of air. It is also generally accepted that the friction is in direct proportion to the length of the pipe, the friction for 200 feet being twice as great as that for 100 feet. The friction is also in proportion to the internal surface of the pipe. Thus a pipe 6 inches in diameter will have twice as much internal surface as a pipe 3 inches in diameter. On the other hand, the 6-inch pipe has four times the sectional area of the 3-inch pipe; so that the interior surface of the 6-inch pipe is proportionally only half as great as the internal surface of the 3-inch pipe. Therefore, other things being equal, the loss due to friction in a 6-inch pipe will be only half that due to friction in a 3-inch pipe.

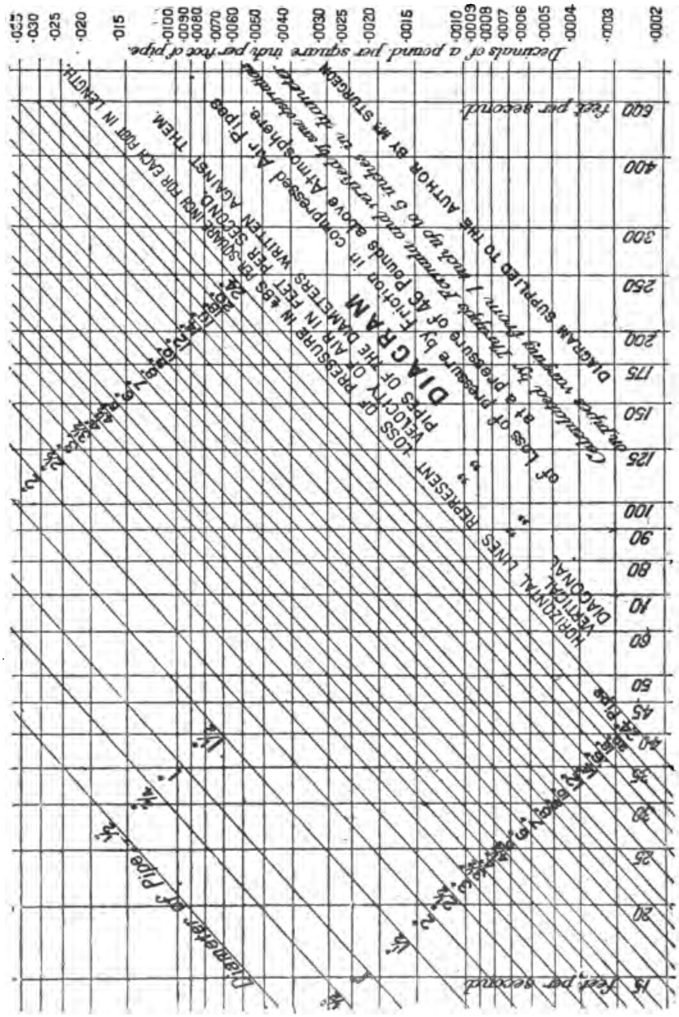


FIG. 483b.—Friction of air in pipes. Horizontal lines, = ; vertical lines, ||.

It is also generally accepted that the amount of friction varies in proportion to the square of the velocities; that is to say, that, comparing the friction of velocity 2 with that of velocity 4, it will be in the ratio of 2^2 to 4^2, or as 4 : 16. Thus for a double velocity the friction is quadrupled, and for a treble velocity the friction is ninefold. It is, however, held by some who have investigated the subject, that the friction does not vary as the square. Mr. Edgar C. Thrupp, as already mentioned in p. 241, as the results of experiments on water, has the following formula :—

Flow of air in wrought-iron pipes—

$$\text{Formula, } v = \frac{R^x}{C\sqrt[n]{S}}$$

v = mean velocity in feet per second.

$$R = \text{hydraulic radius in feet} = \frac{\text{diameter}}{4}$$

$$S = \frac{\text{length of pipe in feet}}{\text{head in feet}}$$

$$\text{For wrought iron} \begin{cases} x = 0{\cdot}65 \\ C = 0{\cdot}004787 \\ n = 1{\cdot}80 \end{cases}$$

If the head is given (as it usually is) in pounds pressure per square inch, this can be turned into head in feet by multiplying the head in pounds by the number of cubic feet of air of the given pressure required to weigh 144 lbs. (that is, equal to 1 lb. per square inch for 1 square foot). Thus, if the pressure is 46 lbs. per square inch above the atmosphere, then 456 cubic feet at 60° will weigh 144 lbs., and if the given head is 0·0002 lb. per square inch, the head in feet = 456 × 0·0002 = 0·0912 feet.

Example.—Air at 46 lbs. pressure.

Loss of pressure per foot run = 0·0002 lb. per square inch.

$$\therefore S = \frac{1}{0{\cdot}0002 \times 456} = 10{\cdot}96$$

$$\text{Log } 10{\cdot}96 = 1{\cdot}0401$$

$$\frac{1{\cdot}0401}{1{\cdot}80} = 0{\cdot}5778 = \log \sqrt[n]{S}$$

$$\text{Log C} = \bar{3}{\cdot}6800$$

$$\text{Log } C\sqrt[n]{S} = \bar{2}{\cdot}2578$$

Diam.	=	2 feet
R	=	0·5 „
Log R	=	$\bar{1}$·699
x	=	0·65
		3495
		4194

$$0{\cdot}699 \times 0{\cdot}65 = 0{\cdot}45435$$

$$\bar{1}{\cdot}0 \times 0{\cdot}65 = -0{\cdot}65$$

$$\text{Log } R^x = \bar{1}{\cdot}80435$$

$$\text{Log } R^x = \bar{1}{\cdot}80435$$

$$\text{Log } C\sqrt[n]{S} = \bar{2}{\cdot}2578$$

$$\text{Log } v = 1{\cdot}54655$$

$$\therefore v = 35{\cdot}2 \text{ feet per second}$$

By means of this formula the loss of pressure in air-pipes for air at a pressure of about four atmospheres has been calculated for various velocities, the results being shown in Fig. 483*b*.

The student will readily understand the use of this table. Suppose, for instance, that it were required to transmit the compressed air to a distance of 5000 feet, with a loss of pressure of 5 lbs. per square inch to overcome the friction of the pipes. Then the required size of the pipe will be found on the line opposite the decimal figure 0·001, which means that the friction per foot of pipe is $\frac{1}{1000}$ part of a pound, and therefore for 5000 feet will be 5 lbs. If the quantity of air is sufficient for say 50 H.P., using 300 cubic feet per H.P. per hour, or 5 cubic feet per H.P. per minute, or a total of 250 cubic feet per minute for the 50 H.P., or rather more than 4 cubic feet per second, it will be found, on reference to the figure, that a 5-inch pipe will take the air at a velocity of about 32 feet per second with the given head, and, the area of this pipe being rather less than $\frac{1}{7}$ of a square foot, the volume delivered will be rather more than 4 cubic feet per second.

This table may be used for air at greater or less densities by altering the right-hand column of figures in proportion. It may also be used for steam by reducing the figures in the right-hand column one half. A further allowance, however, must be made for bends, as per the following table, which shows the loss in pressure for each right-angle bend for air at 46 lbs. pressure:—

TABLE XX.—ADDITIONAL LOSS OF PRESSURE, IN DECIMALS OF A POUND, IN BENDS AND KNEES OF 90° DUE TO CHANGE OF DIRECTION.

Velocities in feet per second.

Velocity..	10	20	30	40	50	60	70
Bends ...	0·0023	0·00092	0·00207	0·00368	0·00575	0·0083	0·0113
Knees ...	0·0033	0·0132	0·0297	0·0628	0·0825	0·119	0·162

Velocity..	80	90	100	125	150	175	200
Bends ...	0·0147	0·0186	0·0230	0·036	0·0517	0·0703	0·093
Knees ..	0·211	0·267	0·33	0·516	0·742	0·866	1·32

From experiments recently made by the author, he is inclined to doubt whether the friction in smooth, straight tubes increases so much with increase of velocity as is generally supposed. Nevertheless, the usually accepted rule is supported by a great body of practical experience.

Hydraulic Power.—In the same way as power is transmitted by pumping air, so it may be transmitted by pumping water. Water, however, is generally used at very high pressures, from 600 to 1000 lbs. pressure per square inch, 800 lbs. being a common pressure. This system is largely adopted on surface works, and to some extent in mines.

Hydraulic pumps in the dip-workings of a mine may be driven by the pressure from the rising main at the pumping-shaft. Water is not so generally convenient for a mine as compressed air. Return-pipes are usually necessary for the exhaust-water. A leakage may cause expense; and great care is required in its use, because of the great momentum of the water moving in the pipes, which causes it to burst the pipes if a valve is suddenly closed, unless there are sufficient safety-valves. The great strength required for the pipes also makes it expensive, except in the case where small pipes, say not exceeding 3 inches in diameter, suffice. In these small pipes there is generally a great margin of strength, which is utilized in the case of hydraulic pressure. The reason why high pressure is used is because water is incompressible, and therefore its density is not increased by increase of pressure, and the friction is generally believed to be proportional to the density. So that there is no more friction in water at 1000 lbs. pressure than at 10 lbs. pressure, whilst the power transmitted by a given weight is, at the higher pressure, one hundred times as great as at the lower pressure. When the transmission of water-power is direct from the generating engine to the motor, say an underground pump, the loss of power may be moderate, say 15 per cent. for the generating engine, and 10 per cent. friction in pipes, leaving 75 per cent. to be utilized by the motor. When, however, the power is used for winding and hauling and various machines, there is great loss, because the pressure of the water is constant, and the power required variable, so that probably only about 25 per cent. is utilized.

Electric Transmission.—Within the last fifteen years the use of electricity has been applied on a considerable scale for the transmission of power in mines. The use of this wonderful agent is one of the greatest triumphs of scientific engineering.

Some instances may be given of what has been actually accomplished. At the St. John's Colliery, Normanton, two coupled engines on the surface, having 22½-inch cylinders and 4-feet stroke, running at fifty revolutions per minute, by means of gearing, belts, and pulleys, give movement to three series-wound Immisch dynamos (see Fig. 484), each about 50 H.P. and capable of giving a current of 60 ampères at 600 volts, and also one small compound dynamo giving a current at 155 volts. The conductors lead from each dynamo down the shaft to a motor in the pit, each dynamo being connected with a motor. The conductors in the

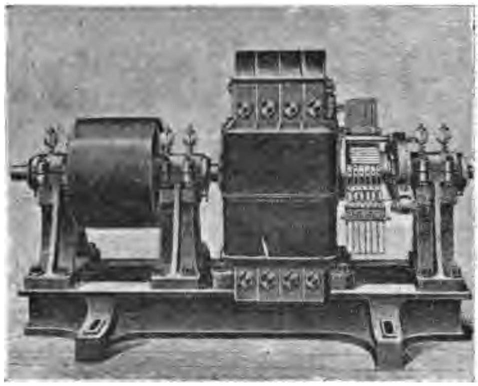


FIG. 484.—Immlsch dynamo.

shaft are similar to those shown at A, Fig. 485, and each have nineteen wires of 16 B.W.G. twisted into a strand and covered with non-conducting composition; this is covered with twisted hemp and fitted into a leaden tube by the Fowler-Waring process. The lead is to keep the conductor dry. Two 50-H.P. motors

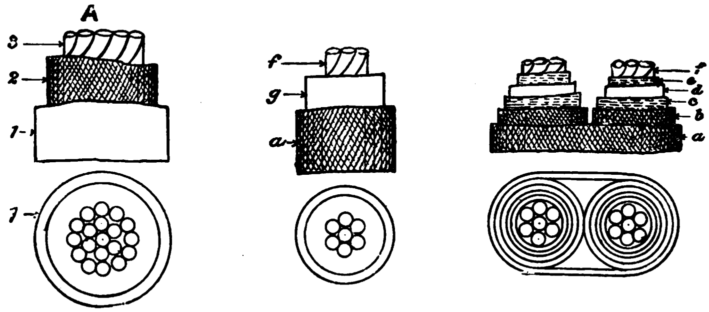


FIG. 485.—Electric cables. A: 1, lead covering; 2, ozokerited hemp braiding and insulation; 3, copper conductor; *a*, waxed cotton braiding; *b*, waxed hemp braiding; *c*, waxed cotton lining; *d*, thin guttapercha; *e*, waxed cotton lining; *f*, copper conductor; *g*, rubber tube.

in the pit, at a distance of 500 yards from the engine on the surface, as measured along the conductors, give movement by spur-gearing and belting to two separate pumps, the horse-power absorbed by each pump-motor being about 35. Another motor of 50 H.P. is attached to a hauling-engine at the same distance from the dynamo on the surface. The power is transmitted to the drum by means of spur-gearing. The total horse-power required for hauling is about 35. Electric conductors are also carried down an incline, and work a 30-H.P. motor and pump at a distance of about 1600 yards from the dynamo, one of the three dynamos of 50 H.P. There are also three pumps, each three-throw, and each driven by a 3-H.P. motor at distances of 1300, 1400, and 2200 yards respectively from the compound dynamo. These are switched on or off as required.

At the Trafalgar Colliery, in the Forest of Dean, a pump is worked by an electric motor, which is at a distance of 1650 yards from the pit-bottom, the horse-power being about 15. At the Pleasley Colliery, near Mansfield, the hauling-engine is driven by electricity near the pit-bottom, at a distance of 725 yards from the dynamo on the surface. Total power required for this engine is about 35·4 H.P. At a large colliery in Glamorganshire a similar system is adopted. At the Cannock and Rugeley Colliery electric haulage has been adopted to the extent of 70 H.P.; and also at the West Cannock Colliery there is an electric motor at a distance of 1400 yards from the shaft. The dynamo on the surface is capable of giving 48 H.P., in the shape of 65 ampères at a voltage of 550. The conductor is a strand of nineteen copper wires, No. 17 S.W.G. At the Shirland Colliery a motor is working at the bottom of a dip, pumping water at a distance of about a mile from the dynamo.

In dealing with electricity, the colliery manager must bear in mind that he has an element capable of producing fire. The two conductors must be placed as far apart as possible, and perfect insulation is necessary. Any failure of insulation may cause what is called a short circuit—that is to say, the electricity, instead of going to the motor and back the proper way, may leap across from one conductor to the other and return by a shorter road. This may cause great heating of the conductors, as well as fire at the point where the leakage occurs. There are several devices for stopping this short circuit, such as fusible conductors at the dynamo, which melt as soon as the current is greater than it ought to be.

It must also be remembered that the spark caused by a severance of an electrical conductor, or by the contact between two bare conductors, might ignite gas. There are apt to be sparks at the motor, which, of course, are inadmissible in any part of the mine where safety-lamps are required, unless the motor is under the continual supervision of a man to test the atmosphere in the same way as if he were going to fire a shot.

There are various contrivances to render sparking innocuous. One of these is Davis and Stokes's patent (see Fig. 486). In this case the commutator, which is the place where sparks are generally produced, has an internal surface instead of an external surface, as usual, and the brushes (A, A), or wire conductors that convey the electricity to the commutator, are inside. The commutator is covered up by an iron case (B B), within which it revolves, but the air-passages up which air or gas could enter the chamber are so narrow that, if the gas entered and fired, the flame would not flash out. At the same time, the entry of gas is considered to be improbable. Some people consider that this arrangement is quite safe.

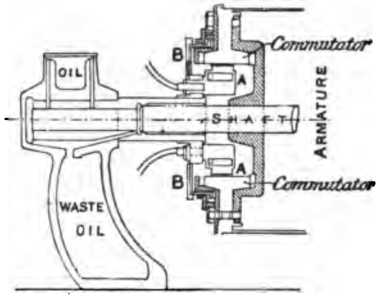


FIG. 486 —Davis and Stokes's safety-commutator.

There are other contrivances having a similar effect. It is probable, however, that the prudent colliery manager will not introduce an electric motor into any part of a mine where the presence of explosive gas is a reasonable contingency. Many engineers, moreover, consider that electricity introduces a danger of setting fire to timber or brattice cloth, and object to its use on that account.

There is no doubt that a fire in a mine is a terrible catastrophe, as the recent accident at Przibram, in Bohemia, a metalliferous mine where there is no fire-damp, is sufficient to prove. When the timber once catches fire, the current of air causes the flame to spread with immense rapidity, and in a few minutes the smoke may become sufficient to suffocate all the men in the return air-road, whilst the flames make the intake air-road impassable.

In dealing with electricity, the question of pressure or tension is very important. The pressure or tension is measured in volts in the same way as the pressure of steam is measured in pounds. For working very small motors of 1 or 2 H.P. a low voltage is required, say not exceeding 100 volts. For a larger horse-power of say 10 or 20, a voltage of 200 or 300 may be used, and for still larger motors a voltage of 500 or 600 is preferred. The higher the voltage the greater the risk of leakage through the conductors, and, consequently, of fire. The lower the voltage the greater the cost of the conductors for a given power.

At the present time, the voltage put into mines, so far as the writer is aware, in this country does not exceed 600. Where the power is to be transmitted on the surface a great distance, as, for instance, from a waterfall to some distant mine, a very high voltage may be safely employed, if it is reduced before entering the mine to a low voltage. A remarkable example of what may be done was exhibited in the Frankfort Exhibition of 1891. In this case the source of power was at Laufen, on the river Neckar. A 300-H.P. turbine (see Fig. 487) gave movement to a dynamo, which produced electricity of 50 volts and 4200 ampères, or about 280 H.P. This passed through three copper rods, $1\frac{1}{8}$ inch in diameter, into a transformer, which is a machine without moving parts, consisting of thick copper wires, or conductors, and thin copper wires. The current, passing through the thick conductors at a low voltage, induces a current in the thin conductors of a

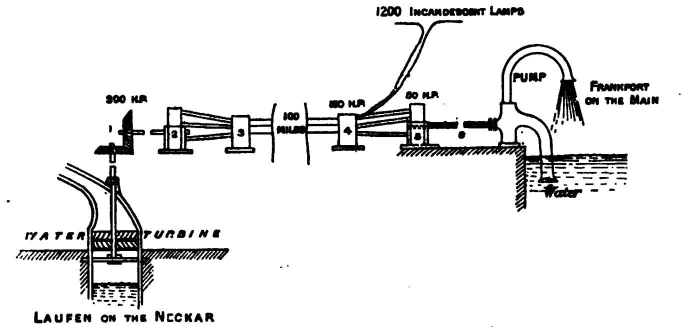


FIG. 487.—Laufen-Frankfort. 1, bevel-gearing from turbine 300 H.P.; 2, dynamo; 3, 4, transformers; 5, motor; 6, shafting to pump.

high voltage, in this case equal to 17,000 volts, at a velocity of 11 ampères. Three copper wires, $\frac{1}{6}$ inch in diameter, pass from the transformer along telegraph-posts to Frankfort, a distance of over 100 miles. At Frankfort the three thin conductors enter a transformer, from which the electricity issues at a voltage of 60 by three copper rods $1\frac{1}{8}$ inch in diameter. The power developed by the electricity at Frankfort in working pumps and in lighting was equal to about 140 or 150 H.P. This is by far the most wonderful achievement in electrical transmission of power. The use of the small conductors reduces the capital cost to a small figure.

The current used is not the ordinary current which has been used in the electrical motors already described. In these motors a "continuous" current is employed, and they are "series" wound. The "series" wound motor has hitherto been considered the most useful for mining purposes. But the current used in the instance given at Frankfort was an "alternating" current, for which the ordinary motor is unsuited, and in the utilization of which for motive purposes there are many practical difficulties to be overcome. In the instance given above, the motor is driven by the "drehstrom," or "twisting current" system, which is one of the more recent inventions of electrical engineers.

Fig. 488 is a diagram intended to give some idea of the action of this current. It will be seen that the three conductors are connected to three equidistant parts of a circle. The three currents are each constantly varying in intensity, and arrive at their maximum intensity in succession. Thus No. 1 current produces the maximum magnetization first, causing the internal armature to revolve towards it; No. 2 then arrives at its maximum intensity, and continues to attract the revolving armature; then No. 3; then No. 1 again. Thus the armature is attracted successively in three different parts, one part being always in proximity to the magnets. In this way a uniform movement is given to the armature.

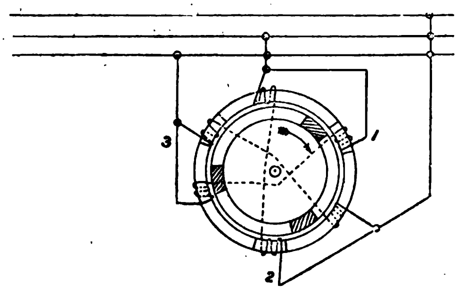


FIG. 488.—Three-current alternating three-phase motor.

The inventor claims that he can make motors of the smallest size to work on this system, whilst, with motors of a moderate size, he has strong bars of copper in their construction instead of small wires such as are sometimes used, and that his machine is therefore more durable.

There is no danger of the ordinary kind of sparking, because the brushes each work on a continuous brass ring, instead of alternately on brass and vulcanite or mica, as in the ordinary commutator; it is the jump from the brass to the brush when the vulcanite intervenes that produces sparks sometimes. In the "drehstrom" motor there seems to be no spark except on reversing the direction of movement.

According to results obtained at Trafalgar Colliery, St. John's Colliery, and other collieries of which the writer has information, the useful effect obtained by electrical transmission, if measured in work done, is about 45 per cent.; but if, instead of taking the work done, the power is taken as delivered into the motor (comparable to the indicated power of an air-motor), then the useful effect is about 55 per cent. This is for a short transmission of say 500 yards. The power absorbed by the conductor in electrical transmissions depends on the voltage and on the size of the conductor, and may be reduced to the minimum by sufficient capital outlay.

The conveniences of electricity are, firstly, the small amount of space occupied by the motor. Owing to the high speed at which it works, a large power can be taken off a small pulley by means of a belting. Secondly, the ease with which conductors can be fixed and taken round corners. With conductors properly wound upon drums, a mile or more can be easily fixed in an afternoon. There are no joints to be affected by moving ground. Also the space occupied by the conductors is very small, so that if additional power is required, additional conductors can be laid without difficulty. In the same way, in the production of electricity, the dynamo can be driven at a great number of revolutions, and a small engine driving it gives out a great horse-power, and therefore the outlay on engines is reduced to a minimum.

For the convenience of those who are not familiar with electrical terminology, it may be useful to give a few definitions. The speed or amount of electrical current is measured in ampères, in the same way as the speed of an air-current is measured in cubic feet, or that of a water-current in gallons or feet per second. The tension or pressure of an electrical current is measured in volts, in the same way as the pressure of steam is measured in pounds per square inch.

The amount of work done is measured in ergs, in the same way as the work done by a steam-engine is measured in foot-pounds. The power of a machine is measured in ergs per second, in the same way as the power of a steam-engine is measured in foot-pounds per minute.

Electricians have three fundamental units of measurement: The *centimetre* (for length); the *gramme* (for mass); the *second* (for time); forming the C.G.S. system.

The resistance of any conductor to the passage of electricity is measured in ohms. The legal ohm is the resistance of a column of pure mercury 1 square millimetre in section and 106 centimetres long at the temperature of 32° Fahr.

The megohm equals 1 million ohms. The microhm equals one-millionth ohm. The legal volt is the electromotive force which maintains a current of 1 ampère in a conductor whose resistance is the legal ohm.

The electrical horse-power equals 746 watts. A watt is the power conveyed by a current of 1 ampère through a conductor by a difference of pressure (*i.e.* potential, or "voltage") between the two ends of the conductor equal to 1 volt; or, in other words, a machine is working at one watt when an ampère passes through an ohm.

By way of illustrating the above nomenclature, we may state as follows: The current of 1 ampère at a tension of 1 volt produces a power of 1 watt, and since there are 746 watts in a horse-power, a current of 1 ampère at a tension of 746 volts gives a power of 1 H.P., or a current of 746 ampères at a tension of 1 volt gives 1 H.P. A current of 74·6 ampères at a pressure of 10 volts equals 1 H.P. A current of 7·46 ampères at a tension of 100 volts equals 1 H.P. Roughly speaking, a current of $7\frac{1}{2}$ ampères at 100 volts equals 1 H.P. Thus a current of 75 ampères at 100 volts equals 10 H.P. A current of 75 ampères at 500 volts equals 50 H.P. For electric lighting with incandescent lamps, a tension of from 50 to 100 volts is usually employed.

A Board-of-Trade unit equals 1000 volt-ampère hours, or 1000 watt hours; thus 10 ampères at 100 volts for one hour equals one Board-of-Trade unit, or a B.T. unit equals 1·34 H.P. for one hour.

A wire of given size will take a given number of ampères; as, for instance, a conductor of 19 wires, each wire 16 B.W.G., is suitable for say 50 ampères for a distance of 1000 yards, with a loss in the conductors of say 10 per cent. of the power delivered. If, instead of 1000 yards, the conductor was 2000 yards in length, it would be necessary to increase its weight per yard to maintain the same useful effect at the far end in the ratio of 1 to 1·414; and if it were 3000 yards, the weight of the conductor would have to be increased in the ratio of 1 to 1·732. Thus it seems that, for a given voltage and a given percentage of loss in the conductor, the weight of the conductor per yard varies as the square root of its length; hence the need of high tensions for economical long-distance transmission by electricity.

If the voltage of the above current is 600, the power transmitted in watts $= 50 \times 600 = 30{,}000$, and in H.P. $= 30{,}000 \div 746 = 40$ H.P.; but, so far as the wire is concerned, it is immaterial whether the voltage is 600 or 6000 or only 60. At 6000 volts the power might be 400 H.P., and at 60 volts the power would be only 4 H.P. In this respect electricity resembles water, the friction of which, in passing through a pipe, is the same at all pressures. Thus in the case of the Laufen-Frankfort transmission, the voltage was 17,000. It is evident, therefore, that for economical electrical transmission a high voltage must be employed. Against this must be set the dangers and difficulties of dealing with a high voltage underground. The following statement shows what may be expected in a first-class electrical installation:—

Generating steam-engine	100 H.P.
Friction of engine, belts, etc.	10 ,,
Loss in dynamo, say	13 ,,
Loss in conductor one mile	12 ,,
Loss in motor	10 ,,
Net power of machine, whether pump, hauling, drum, or other ...	55 ,,

Within the last five years the use of electricity has made very great advances. In some of the mining districts of Australia, America, etc., central electrical plants have been erected for the supply of power to mines in the district; large electrical power-distribution plants have been constructed in Switzerland and other European countries; and large schemes of a similar kind are now being actively promoted in this country. Electrical transmission of power to underground workings is now becoming general, and the dynamos and motors are much improved. The ordinary commutator now works without apparent injury from sparking or otherwise, and seems, when well looked after, likely to endure for an unlimited period. Multiphase currents are also coming into general use.

Rods.—Power can be transmitted to great distances by rods from one oscillating lever to another, and in this way has in some

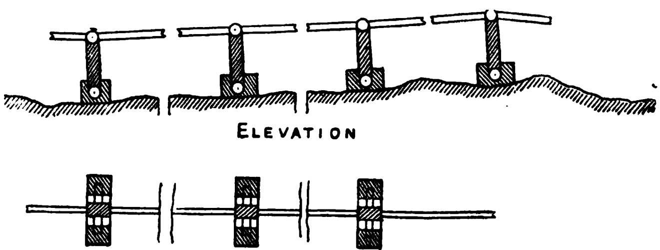


FIG. 489.—Transmission by rods.[1]

cases been carried for miles over hill and dale, say from a water-wheel to some distant mine, as illustrated in Fig. 489. This method of transmission is commonly used in the case of pump-rods. Sometimes a length of half a mile of rods is taken from some main steam-engine down an incline. Revolving shafting is sometimes used on the surface for transmitting the power of an engine for a distance of several hundred feet.

Wire-rope Transmission.—This is one of the chief systems of transmission employed by mining engineers, and under certain circumstances is the most economical method that can be adopted. The writer has seen the power of a steam-engine on the surface transmitted to an ore-dressing plant at a distance of several hundred yards by means of a light steel rope $\frac{9}{16}$ inch diameter, driven by a pulley 16 feet diameter on the steam-engine, the rope making simply half a turn on this pulley, the bottom of the groove being lined with indiarubber so as to save the rope

[1] In this figure the rods are shown broken between each vibrating lever, merely to indicate that the length of horizontal rod is greater than can be shown in this scale.

FIG 490.

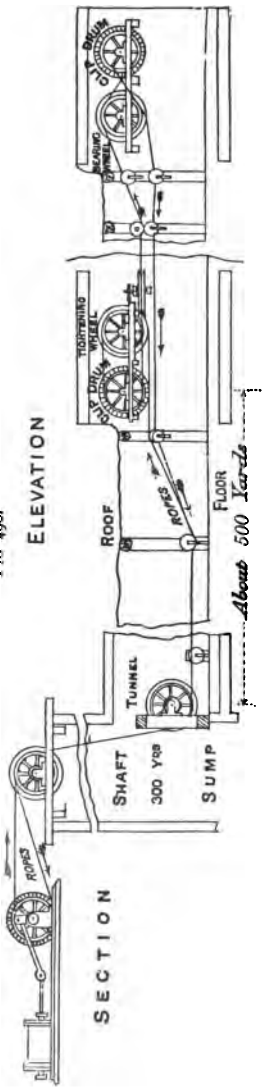
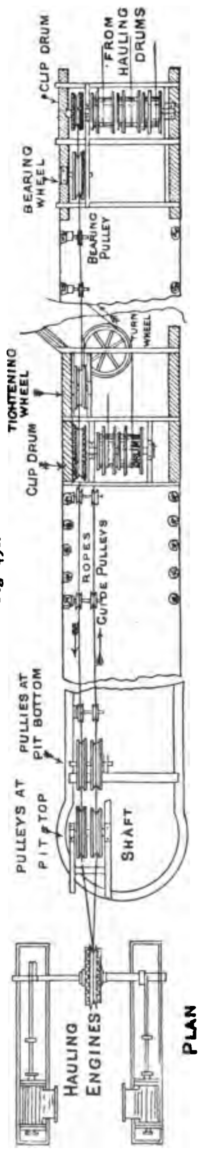


Fig. 491.



FIGS. 490, 491.—Transmission of power by endless wire rope from engines on surface to haulage-drums at inbye end of level, the drums hauling up steep inclines by separate ropes.

from unnecessary abrasion; a similar pulley at the machinery-house received the power, about 100 H.P. Owing to the great speed at which the rope travelled—one mile a minute—the tension was comparatively small; there were no pulleys to absorb any power in friction, the rope being stretched in mid-air. In this case the loss of power would be no more than in transmitting from one pulley to another in the same room. In underground transmission the conditions are seldom so simple; a good many pulleys are required, and turn-wheels, and a large percentage of the power is thus frequently lost. Figs. 490 and 491 give an example of a rope-transmission plant erected by the writer a good many years ago. In this case a pair of engines on the surface, 25-inch cylinders, 4-feet stroke, drove a 6-feet Fowler's clip-drum, round which passed a $\frac{7}{8}$-inch endless steel rope. The ropes passed over two pulleys on the pit-frame, and underneath two pulleys at the pit-bottom about 300 yards below, then round two horizontal pulleys, then carried on rollers along a straight road. At a distance of about 500 yards from the pit-bottom, it passed round a guide-wheel and a 4-feet clip-pulley connected by clutches with two drums, and then at a distance of about 600 yards it passed round another pulley, on a shaft on which were three drums connected by clutches as required for hauling up three inclines. This is a typical system of wire-rope transmission. The loss of power depends upon the care exercised in making, adjusting, and lubricating the pulleys and rollers. It frequently happens in practice that 50 per cent. of the power is lost in this way; but the loss is generally due to want of sufficient care in the construction and maintenance of the machinery. A considerable amount of power is also lost in bending the stiff wire rope round the wheels, a loss which is repeated, of course, at every wheel round which the rope is bent, the amount of this loss being inversely as the diameter of the wheel (but not exactly proportional to this ratio), and irrespective of the angle of the bend.

Gas and Oil Engines.—Gas-engines can be conveniently used on the surface, but are unsuitable for mines, owing to the danger from escaping gas. This difficulty is got over by the use of the oil-engine. This is driven by the explosion of paraffin-oil spray mixed with air, in the same way as gas-engines are driven. To start the engine a large lamp or a fire is required, which is placed inside the base of the machine. After the engine has been started, the heat of the explosion in the cylinder is sufficient to warm and evaporize the injected oil, and there appears to be no exposed flame or heat capable of igniting fire-damp; but in using this engine it must always be borne in mind that fire is required every time the engine is started. The oil-engine is now used in many mines for pumping, hauling, and rock-drilling.

CHAPTER XVIII.

WINDING - ENGINES : VERTICAL, HORIZONTAL, COUPLED ; SIZE, POWER, SPEED ; VALVES, BRAKES, AUTOMATIC STOP-VALVES AND BRAKES, BALANCING, ETC.

WINDING-ENGINES are the machines by which the mineral and sometimes the workmen are raised from the mine. One of the simplest and most primitive is the windlass (see Fig. 492). This may be used with one rope and one bucket, or with two ropes and buckets, or with one rope turned seven or eight times round the barrel and two buckets, but this is only suitable for a very

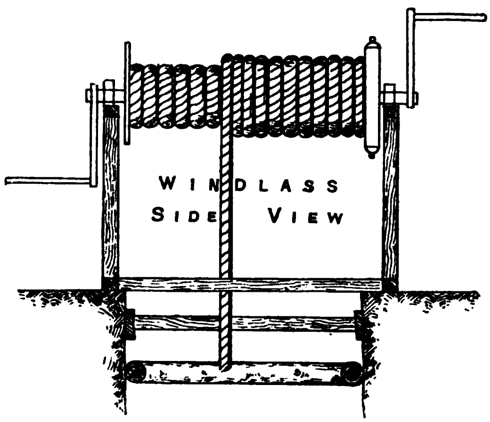


FIG. 492.—Windlass: side view.

slight depth. The windlass may be constructed with a saddle made of iron bars (see Fig. 493). The rope is wrapped round ten or twelve times, and at each end is hung a bucket; as the windlass is turned, the friction of the rope raises the loaded bucket, but the shape of the saddle prevents the rope from screwing itself off the end. Horse-power is often applied. One simple means is shown in Fig. 213; another is a horse-gin, or drum turned by a horse (see Fig. 471). Horse-gins are seldom used nowadays.

Water-wheel. — A water-wheel is often used for raising the cage or bucket (see Figs. 494 and 494*a*). The empty cage is lowered down by disconnecting the winding-drum from the wheel and applying a brake. In other cases the motion of the drum is reversed by means of changeable spur or bevel wheels, connecting the shaft of the water-wheel with the drum-shaft. In Germany a double water-wheel is used (see Fig. 495), the buckets of one wheel fixed to turn in one direction, and the buckets of the other fixed to turn in the other direction. Valves turn the water-

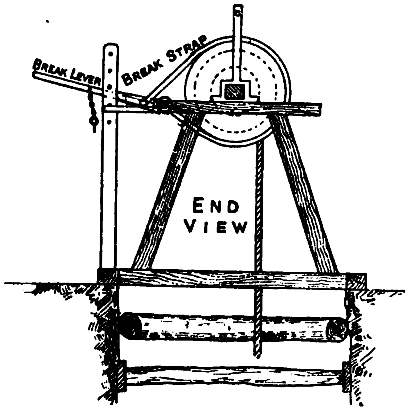


FIG. 492*a*.—Windlass: end view.

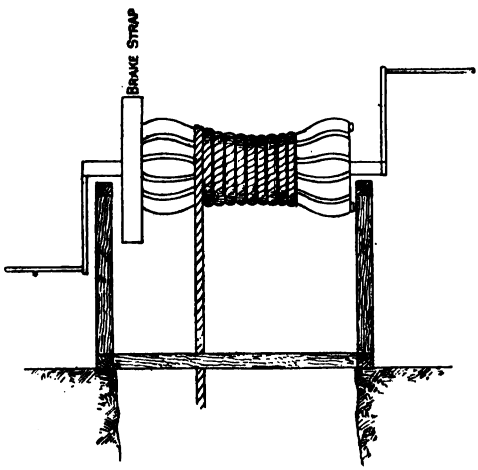


FIG. 493.—Windlass: iron saddle.

current on to one wheel or the other, according to the direction in which the drum has to be turned.

Water-balance.—A method that has been used in some of the Glamorganshire mines is a water-balance (see Fig. 496). In this case there are two cages; underneath the bottom of each is

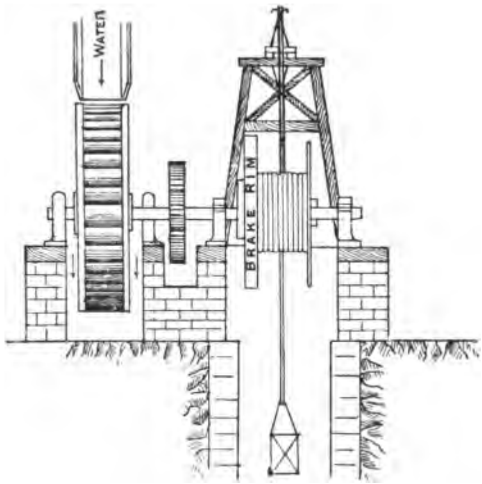


FIG. 494.—Single water-wheel.

fixed a water-tank. The cage tank which is at the top is filled with water from a spout, the weight of which overbalances the load of coals in the other cage, which is so raised to the surface.

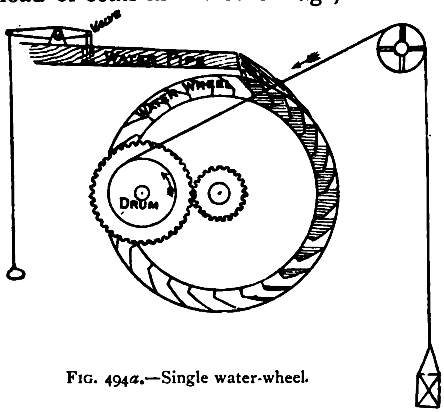


FIG. 494*a*.—Single water-wheel.

On reaching the bottom, a valve in the bottom of the water-tank is forced open by a block of wood pressing against the valve-spindle, and the water escapes. The water has to be pumped up by a pump worked by the water-wheel, unless there is an adit on the lower level by which it can escape. This method of winding is slow.

Steam-engine. — The steam-engine is now almost universally used. One hundred years ago the kind of engine employed was that now known as the atmospheric engine. The top of the cylinder is open, and

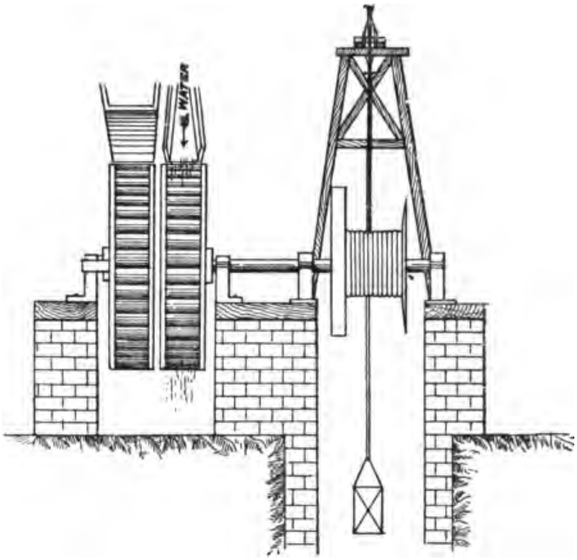


FIG. 495.—Double water-wheel.

the top of the piston is subjected to the pressure of the atmosphere. By means of the condenser, the pressure of the atmosphere is removed from the underside of the piston, which thus makes the down-stroke. The up-stroke of the piston is made by the heavy weight on the connecting-rod at the outer end of the beam. Engines of this class have been frequently seen at work within the last thirty years. The Boulton and Watt engine has a cover on the cylinder; the condenser is removed to a distance of a few feet from the cylinder. The engine is generally double-acting; that is to say, steam is admitted both above and below the piston. The winding-drum in the atmospheric engine and in the Boulton and Watt engine was on the second motion; that is to say, the drum was on a shaft separate from the engine-shaft, the movement being com-

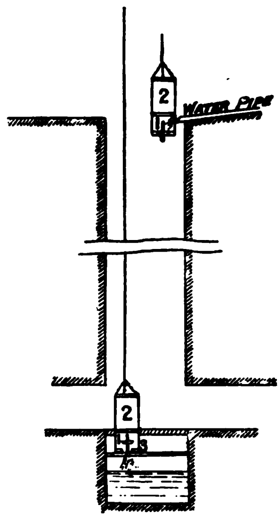


FIG. 496.—Water-balance. 1, water-compartment and valve; 2, cage for tubs; 3, valve lifted automatically at bottom.

municated by spur-wheels, that on the engine-shaft being say one-third of the diameter of the other on the drum-shaft, thus reducing the speed of the drum. In more modern times the drum was placed on the engine-shaft, necessitating a larger engine to make the winding quicker. Winding-drums are often on the second motion at the present time, when only a slow speed of winding is required. Beam-engines are sometimes, but rarely, used for winding at the present day.

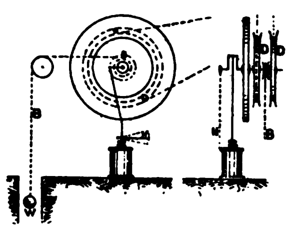


Fig. 497. Fig. 498.

FIG. 497.—Single vertical engine: side view. FIG. 498.—Ditto: end view.

Vertical direct-acting single engines came largely into fashion, and between thirty and fifty years ago were probably the favourite type. An example is shown in Figs. 497 and 498. Such an engine as there shown would have a cylinder say 30 inches in diameter, with a 5-feet stroke, and a rope-roll say 10 feet in diameter, or it might be of larger size, say 40-inch cylinder and 6-feet stroke, and a rope-roll 12 to 14 feet in diameter. The ropes used on the engine shown in

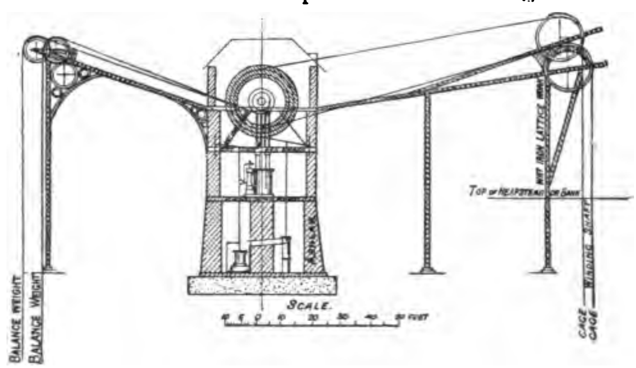


FIG. 499.—Wilson winding-engine: single cylinder, condensing, flat-rope drums.

the figures being a flat iron wire rope, the diameter of the rope-roll increases as the rope is wound up. There was a heavy fly-wheel on the shaft, to which a brake-strap was applied.

Fig. 499 shows a large engine erected at Monkwearmouth Colliery about twenty-five years ago. The winding-engine is direct-acting, and has a single vertical cylinder 68 inches in diameter, with a 7-feet stroke. The boiler-pressure is 20 lbs.; there is a condenser in which the vacuum reaches 12½ lbs. The steam-valves are double-beat Cornish 15 inches in diameter; the exhaust-valves are the same—they lift 1⅛ inch. The drums for flat ropes are 22 feet in diameter. When the rope is wound up, the diameter is 25 feet; 23¾ revolutions of the drum raise the cage from the bottom to the top. The cage is of steel, with four decks; each deck has two tubs or waggons; each waggon weighs about 5½ cwt., and holds about 8½ cwt. of coal. The time occupied in each journey is about 1½ minute; it takes from 8 to 10 seconds to run the full tubs off and empty tubs on to each tier of the cage, and 6 seconds to start at the beginning of each journey. There are six boilers, each 33 feet long and 7½ feet in diameter.[1] The piston-rod of the engine is guided in a strictly vertical direction by means of a parallel motion consisting of two beams and a connecting link fastened to the cross-head, as shown in the figure, and on somewhat larger scale in Fig. 500. In this figure E

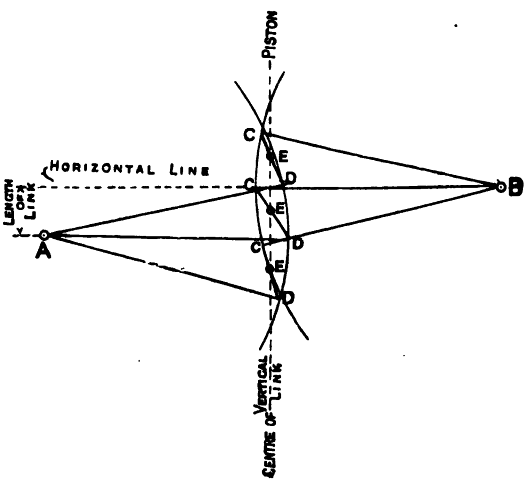


FIG. 500.—Parallel motion. A, B, fixed ends of levers; C, D, E, various positions of link; E is the pin passing through cross-head connecting rod and link; C is the pin connecting lever B to the link; D is the pin connecting lever A to the link.

represents a pin through the cross-head of the piston-rod, to which is attached a link C D, which is free to revolve on the pin E; at C is a pin connecting the link C D with a strong lever or beam C D; at D is a similar pin connecting the link C D with a similar strong lever or beam A D. The beam C B turns on the centre B, and the beam D A turns on the centre A. It will be

[1] These particulars were taken in the year 1870.

seen that the centre B is on a level higher than the centre A by a height nearly equal to the length of the link C D. If we assume that the piston is at half-stroke, then the pin E will be in the middle of the three places denoted by the letter E on the vertical dotted line. If the piston moves upward, the tendency of the lever B C is to pull the cross-head towards B, whilst the tendency of the lever D A is to pull the cross-head towards A. These two tendencies counteract each other, and the movement of the piston is thus maintained in a vertical line. In many vertical engines the piston-rod is guided by bars of cast iron, between which the cross-head slides; there is, of course, friction between the slide and the bars, and considerable care has to be taken in their adjustment, in order that they shall not be so tight as to cause undue friction, nor so loose as to allow perceptible oscillation of the piston-rod. The levers used for parallel motion are used for working the air-pump rod, cold-water pump rod, valve rod, etc.

Coupled Engines.—Winding-engines are now almost invariably made in pairs with a crank at each end of the drum-shaft, as in Fig. 501, which represents a pair of horizontal winding-engines, with cylinders 36 inches in diameter, 6-feet stroke, rope-rolls being about 14 feet in diameter, the fly-wheel serving as a brake rim in the centre, flat iron or steel wire ropes being used. The valves, being equilibrium or double beat, are so adjusted that the engine-man can easily reverse or moderate the engine by means of the link motion. The cranks are placed at right angles, so that both engines can never be at the dead-centre at once. With a single engine it is evident that if the engine is standing, and the centre of the crank-shaft and the piston-rod are in the same straight line, no amount of pressure on the piston will move the engine, and some external force has to be applied to the drum to start it; with the coupled engines, when one is on the dead-centre, the other is at right angles, and its piston has the maximum leverage for lifting the load. These engines can therefore be started or stopped at any part of the stroke, and the starting or stopping can be made very gradual; in this respect they have a great advantage over the single engine, which must be always brought to rest when the crank is nearly at right angles to the direction of the piston-rod, and a sudden start must be made in order to get over the dead-centre. For these reasons coupled engines are now universally employed for winding, the sectional area of the two cylinders being together equal to the sectional area of the single-cylinder

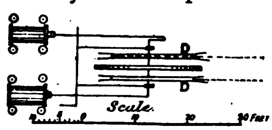
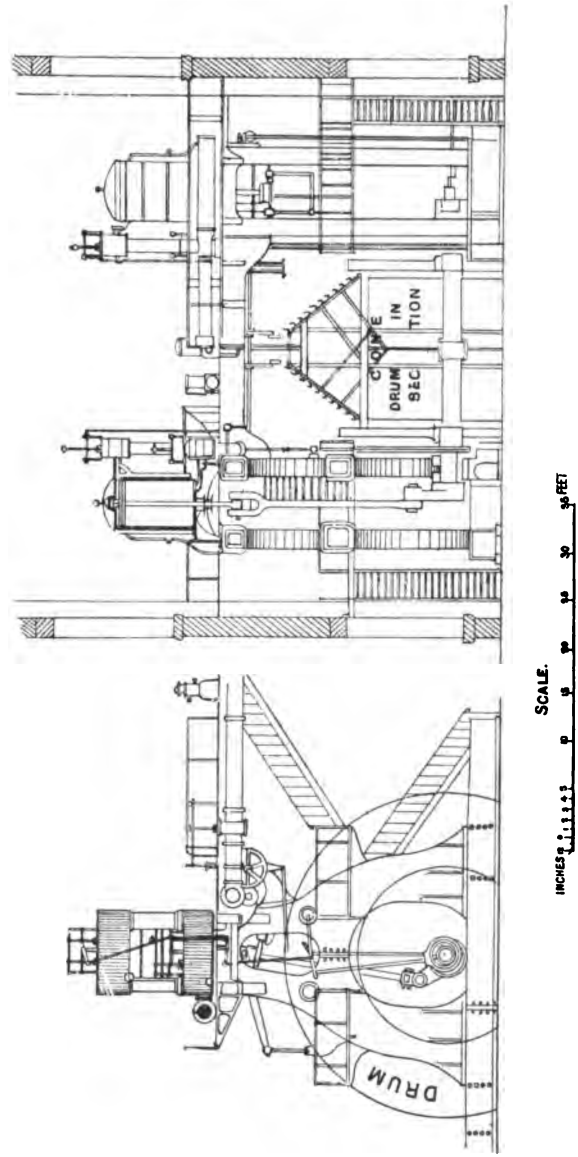


FIG. 501.—Coupled engines: cranks at right angles.



FIGS. 502, 503.—Coupled engines: inverted (Harris's Navigation).

engine, supposing the stroke to be of equal length in each case.

Inverted-cylinder Engines.—Figs. 502 and 503 show a pair of inverted cylinder winding-engines, the drum-shaft being near the ground-level, and the cylinder above on an iron frame. The engines here represented were those erected at the Harris's Navigation, in Glamorganshire. Each cylinder is 54 inches in diameter; the stroke is 7 feet; the cylinders are steam-jacketed; the valves are double-beat, steam-valves 14 inches in diameter; exhaust-valves, 16 inches; trip expansion gear is applied to the steam-valves. The cranks are of hammered iron; the crank-pin is 11 inches in diameter; the shaft is of wrought iron, 24 inches in diameter. Bearings are 20 inches in diameter by 30 inches long.[1] The drum is a scroll or spiral drum, 18 feet on the small diameter, rising by fourteen coils on the scroll to a diameter of 32 feet. The cages and bridles were each 2 tons 10 cwt., holding four trams weighing 2 tons, containing coal 6 tons; total, 10 tons 10 cwt.

Horizontal Engines.—It is now usual to lay the engines

Fig. 504.

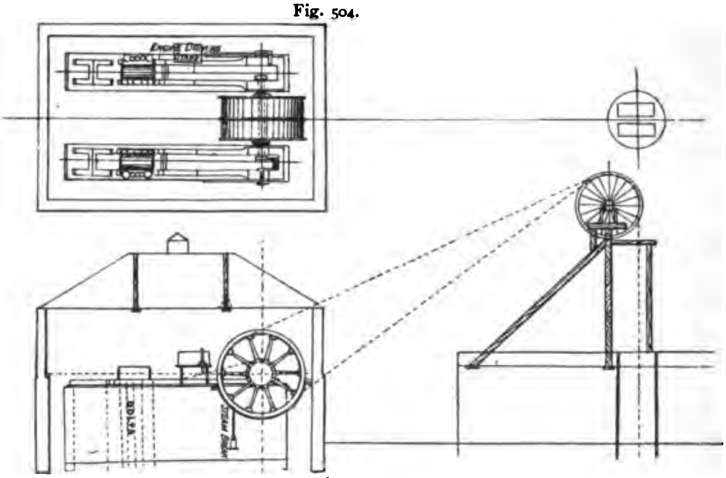


Fig. 505.

FIGS. 504, 505.—Coupled engines: horizontal (Denaby).

horizontal on a bed of masonry, the cast-iron bed-plate being bolted down to the masonry, the cylinders, guide-bars, and pedestals bolted on to the cast-iron bed-plate (see Figs. 504,

[1] Above particulars and the figures were taken from a paper for the Institution of Civil Engineers, by Messrs. T. Forster Brown and G. F. Adams, on deep winning of coal (vol. 64, part 2).

505). The horizontal and vertical engines each have their advocates, and it is almost impossible to say that one class is better than the other. The engines shown in the figure were examined by the writer more than twenty years ago, and still continue in full work, though the machinery has since been modernized by the entire substitution of steel for iron in the ropes and cages, and in other ways.

TABLE XXI.—TABLE OF CHIEF DIMENSIONS, WEIGHTS, ETC., OF THE WINDING-ENGINE (in 1870).

Description of engine:	horizontal high-pressure engine.
Number of cylinders:	two.
Diameter of cylinders:	40 inches.
Effective area of cylinders together:	2490 square inches.
Length of stroke:	6 feet.
Pressure of steam:	42 lbs. maximum observed in cylinders; gauge about 45 lbs.
Average vacuum in condenser:	no condenser.
Maximum velocity of piston:	700 feet a minute.
Mean ,, ,,	360 ,, ,,
Description of valves:	double-beat brass valves.
Diameter of steam-valves:	about 13 inches.
Diameter of exhaust-valves:	about 15 inches.
Lift of valves:	$1\frac{1}{2}$ and $1\frac{7}{8}$ inch.
Area of steam port:	about 80 square inches; the same port serves for steam and exhaust
Ratio of area of port to area of cylinder:	1 : 16.
Diameter of steam-pipe:	$9\frac{1}{2}$ inches branch.
Diameter of exhaust-pipe:	10 inches.
Length of exhaust-pipe:	70 feet to feed water-heater with seven elbows from right-hand engine. The other exhaust-pipe is shorter.
Number of exhaust mains:	one.
Maximum diameter of drums:	18 feet.
Minimum ,, ,,	18 feet.
Mean ,, ,,	18 feet.
Revolutions per journey:	$23\frac{3}{4}$.
Mean revolutions per minute.	30.
Weight of drums and fly-wheel:	about 73,000 lbs.
Weight of shaft:	about 17,000 lbs.
Total weight of moving parts, including load, etc., moving at a diameter of 18 feet, all weights at a greater or less diameter reduced to their value at the said diameter:	80,000 lbs.; equals factor of energy for any given velocity.
Total weight of moving parts, including ropes, pulleys, coals, drums, and engine:	about 158,000 lbs.
Description of ropes:	round wire ropes, one iron and one steel.
Weight per fathom:	iron 29 lbs., steel 23 lbs.

Weight of cage:	iron cage, 5376 lbs.
Weight of tubs (four):	about 2130 lbs.
Weight of coals:	4480 lbs.
Ratio of gross to net load:	2·67 : 1.
Mean speed of cage in shaft:	1691 feet per minute
Maximum speed of cage in shaft:	3080 feet „ „
Depth of pit:	1351 feet.
Time occupied each journey:	47 seconds.
„ „ in landing:	23 seconds.
Foot-pounds of power exerted by the engine each journey:	about 12,000,000 foot-pounds.
Foot-pounds of useful effect or duty each journey:	6,300,000 foot-pounds.
Ratio of gross to effective power:	100 : 53
Ratio of maximum pressure to dead load at commencement:	100 : 50
Boilers, description:	plain, externally fired, egg-ended boilers.
„ number:	eight, seven at work.
„ dimensions:	30 feet long by 5½ feet in diameter, and 6 feet in diameter.
„ heating surface each boiler:	say 200 square feet.
„ grate surface each boiler:	35 square feet.
„ ratio of grate to heating surface:	7 : 40.
Maximum indicated horse-power:	950 H.P.

Valves.—One of the best kind of valves for ordinary engines is the slide-valve. This valve is kept steam-tight against the face of the valve-chest by the pressure of the steam at the back. It is often used for winding-engines, but for engines with cylinders 20 inches and upwards in diameter, with a steam-pressure of say 30 lbs., or for engines of say 18 inches in diameter, with a steam-pressure of 60 lbs. and upwards, a good deal of power is required to overcome the friction of the valve. The amount of this power is of no importance when the work is done by the engine, but when the engine is reversed, stopped, and started, the valve has to be worked by the engine-man, who has to move this valve perhaps four or five times a minute. For this reason, when the slide-valve is used for large winding-engines, some contrivance is applied for balancing the pressure of steam so as to reduce the friction of the valve. Several ways of effecting this result are adopted, one of which is shown in Fig. 506. In this case the valve-box cover has a large tube standing out say 2 feet; near the end of this is a thin steel diaphragm, *a*, from which a rod is connected to the back of the slide-valve. The pressure of the steam on the diaphragm, acting through the rod, tends to hold the valve off the face, but, the diaphragm being less in area than the valve, there is sufficient steam-pressure to keep the valve on the face. It is liable to rattle sometimes, when the throttle-valve is closed.

It is, however, a general rule to use the double-beat or equilibrium valve, shown in Fig. 507. It has several advantages over the slide-valve. The distance the valve has to move in reversing the engine is less, say $\frac{1}{3}$ or $\frac{1}{4}$; consequently, the leverage of the engine-man is three or four times as great. The pressure of the steam does not cause any friction in the movement of the valves, other than the friction due to the stuffing-boxes of the valve-

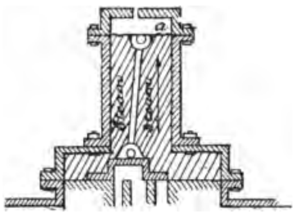


FIG. 506.—Balanced slide-valve.

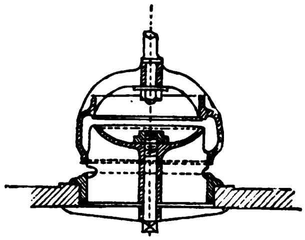


FIG. 507.—Equilibrium-valve.

spindles, which friction has also to be dealt with in the slide-valve. The only power required, other than that necessary to overcome the friction of the joints, links, levers, and of the spindle stuffing-boxes, is that necessary to overcome the pressure of steam upon the valves which are closed. There will be one valve to lift at once for each cylinder, so that the engine-man must have sufficient leverage to overcome the pressure of steam upon two valves. Owing to the construction of the valves, the steam-pressure, however, only takes effect upon two narrow rings round each valve, say $\frac{1}{4}$ inch wide. Supposing the diameter of the valve to be 15 inches inside, the circumference will be about 48 inches, and, being $\frac{1}{4}$ inch wide, this gives 12 square inches of area for each ring, or 24 square inches for both, on which is a steam-pressure of say 60 lbs.; the total pressure keeping the valve closed will thus be 1440 lbs.; as there are two valves to lift at once, the total is 2880 lbs. pressure. If the valves lift a maximum distance of $1\frac{1}{4}$ inch, the lever will have to travel say $2\frac{1}{2}$ inches, and to accomplish that movement the engine-man will move his hand say 50 inches, or twenty times as far; so that 2880 lbs. of weight has to be divided by 20, requiring a pressure of 144 lbs. on the lever-handle. This would be too great a pressure for the man to exert for the whole 4 feet through which he moves his hand-lever; but it has only to be exerted for a small fraction of 1 inch, because the instant the valve is lifted from its seat the steam-pressure gets under it, and the power required to lift it is then comparatively slight, and the force to overcome this weight of 144 lbs. is found in the momentum of the levers and balance weights, which move some distance before the lift of the valve begins.

Steam reversers, which thirty years ago were a novelty and rarity in winding-engines, are now often used, and greatly reduce the labour of the engine-man.

Corliss' valves are now commonly used for winding-engines in places where fuel is expensive, as in the Transvaal, and lately winding-engines with these valves have been erected at Tredegar by Messrs. Markham and Co., of Chesterfield.

Compound engines are used in some cases, and give satisfaction.

Link-motion.—The valves are generally worked by means of eccentrics, and the engine is reversed and regulated by means of a link (see Fig. 508). There are two eccentrics, *a* and *b*, for each cylinder; each eccentric-strap is connected by a rod, A and

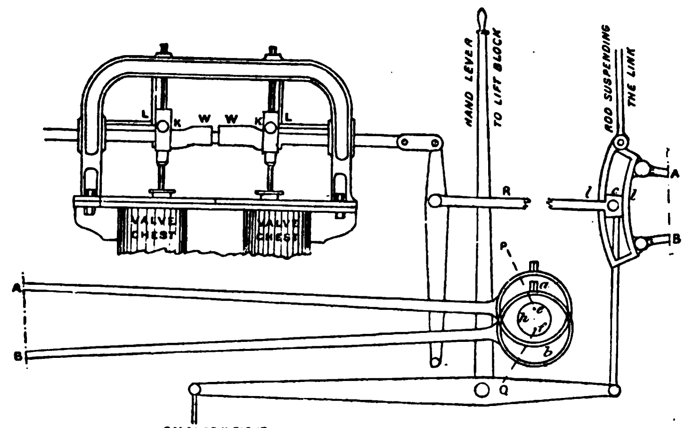


FIG. 508.—Eccentrics, eccentric-rods, link, valve-rods, hand-lever, valve-chests, valve-spindles, etc. This figure only shows two valves out of the four required for each cylinder.

B, with one end of the link, *l*; the valve-rod, *r*, ends in a block, *c*, which will slide up and down in the link. When the block is at the bottom of the link, it is necessarily driven by the eccentric-rod connected to the bottom of the link. If the block is moved to the top of the link, it is then driven by the eccentric-rod connected to the top of the link. If the block is put in the middle of the link, it is not moved at all, as the link turns upon its own centre. If the block is moved halfway between the centre of the link and one end, the valves are then opened only half their maximum distance, and are sooner closed, giving the engine less steam, and thus the speed is regulated. Instead of moving the block up and down in the link, it is a common plan to move the link up and down over the block.

It must be borne in mind that the effect of the link and eccentric is in practice not exactly what it ought to be, and there is a great variety of design in the link and connections with the valves in order to attain a regular and uniform action of the valves. For the economical use of steam and the quick working of the engine, it is necessary to give the valves a little "lead;" that is to say, the steam-valve must begin to open a very little before the beginning of the stroke, and the exhaust-valve must open a little before the end of the stroke, say at $\frac{19}{20}$. This is necessary for two reasons: Firstly, the opening of the valves is gradual, and unless the opening begins a little too soon, the valves will not open in time to their full width. Secondly, even supposing the valves to be opened to their full width, it is desirable that the steam should begin to enter the cylinders a little before the beginning of the stroke, and so cushion the piston; and it is also desirable that the exhaust steam should begin to leave a little before the end of the stroke, and so give the steam time to get out of the cylinder before the piston has advanced far on the return stroke. The effect of want of "lead" is clearly shown in Fig. 518, where, owing to the steam-valve not having sufficient "lead," full steam-pressure is not attained until the piston has completed one-tenth of its stroke, and where, owing to the exhaust-valve not having sufficient "lead," the steam never gets out of the cylinder during the return stroke. This is due to the high speed at which the engine is going at the seventh revolution. On referring to Fig. 519, where the engine is only just starting, it will be seen that, owing to the slow speed at which the piston is moving, there is time for the steam to attain full pressure at the back of the piston, and also for the exhaust steam to escape from the front of the piston. In order to give the valves the requisite "lead," it is necessary that the eccentrics should be put forward. Referring to the Fig. 508, the eccentrics *a* and *b* are shown with their centres *e* and *f* in a vertical line one above the other. This represents them with no "lead." In order to give a "lead" the eccentric *a* will have to be moved round the shaft *h* in the direction of the line *e p;* and the eccentric *b* must also be moved round the shaft in the direction of the line *f q*. This throwing forward the eccentric gives, of course, a corresponding movement to the link, and through the block to the valve-rod and valve. The eccentrics are generally fixed either with a hollow key, so that they can be moved round the shaft to any desired position by slackening the key, or are attached to a fixed disc on the shaft by means of a pin working in a slot in the fixed disc which can be tightened at the required position. The exact amount of "lead" to give a valve requires to be very carefully determined by drawings and models.

In some places the engine-man is saved the labour of working the valves of the main engine by hand, his hand-lever merely giving motion to a small steam-engine (see A, Fig. 509, from a photograph by the author), which raises or lowers the links, B, of the main engine. Whilst the use of this subsidiary engine to work the valves is often found very convenient, the general practice in England is so to balance the valve and valve-gearing that the engine-man is able to reverse the valves with ease, without any

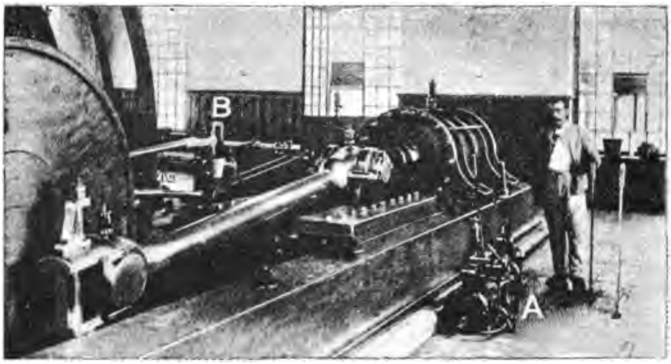


FIG. 509.—Engine-man working valves by hand-levers, actuating small engine that works the main-valves.

other power than that of his own arm. Equilibrium-valves must always be so set that the steam-pressure is on the top of the valve, and therefore the valve cannot be opened by the steam-pressure; thus the steam-valves open against the boiler-pressure and the exhaust-valves open against the cylinder-pressure, so that, no matter how much the steam in the cylinder is compressed, it will not open the exhaust. This is essential to safety, though some engine-makers have failed to understand the necessity for this arrangement.

Instead of lifting the equilibrium-valve spindles by levers, they are sometimes lifted by wedge-shaped slides, as shown in Fig. 508, letters W, W.[1] The effect of these wedge-slides is not merely to open the valves, but to close them, so that there can be no possibility of a valve sticking open. This seems to be a great advantage. In order that these wedge-shaped slides may lift the levers without much friction and without throwing an undue lateral strain

[1] Sketched from engines erected by Messrs. Robert Daglish and Co.

on the valve-spindle, friction-rollers, K K, are fixed on the spindle, against which the wedge or inclined plane presses. The valve-spindle is also guided by a vertical slide, L, to counteract the effect of the lateral thrust of the wedge-block. These wedge-blocks can be moved on the horizontal rod to any position, so as to give the desired lift of the valves and best cut-off of the steam-valve.

Expansion-valves.—There is great economy in working steam expansively; to do this the steam-valve must be opened as wide as possible at the beginning of the stroke, so as to admit steam to the cylinder at the full boiler-pressure. When the piston has travelled from one-quarter to one-half of its stroke, the steam-valve is suddenly closed, the rest of the stroke being accomplished by the expansion of the steam; this may be accomplished, with a double-beat valve, by the use of trip-gear, a lengthened description of which would be out of place in an elementary treatise.

Methods of working winding-engine valves with variable expansion have been devised by Guinotte, Audemar, John Daglish, Robert Daglish, Thornewill and Warham, Melling, and many others. The system now preferred is automatic. On this system the centrifugal force of revolving weights like governor-balls is used to operate the trip-gear. Thus, when the required engine speed has been obtained the cut-off is effected, and at slower speeds the engine has full steam. Melling's gear gives a gradually increasing expansion as the speed increases.

For perfect safety in the use of automatic cut-off gear, there should be an attachment for cutting off the action of the centrifugal force in case the engine valves are reversed at full speed to stop the engine.

Cam-gear.—On the continent it is quite common to work the valves by means of cam-gear instead of link-motion. Fig. 510 shows a method of doing this. By means of bevelled gearing, A, from the drum-shaft, rotary motion is given to a long shaft, B, which passes by the side of each cylinder; on this shaft is a sliding cylinder, C, with cams, two for each valve. A rod from each valve rests upon a cam. As the shaft revolves, the cam raises or lowers the valve-rod, so opening or closing the valve. By sliding this cylinder, the cams are moved from under the valve-rods till the valve-rods rest upon that part of the revolving cylinder where there is no cam, and the valves have thus no movement, but remain closed. The cams are gently tapered, so that, if the sliding cylinder is only moved a little way, the valve-rod rests upon the smaller part of the cam; the steam-valve in this case, instead of being kept open for the whole stroke, is only kept open for say half of the stroke, the exhaust-valve having the same lift as before. The engine is reversed by sliding the cylinder until another set of cams comes under the valve-rods, by which

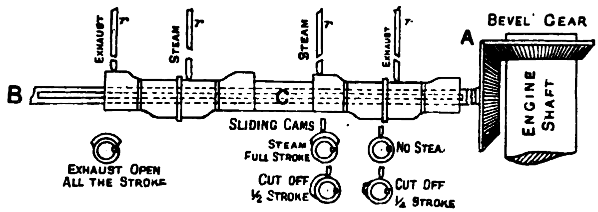


FIG. 510.—Sliding cam-valve gear for winding-engines. *r*, rods to open the valves.

their period of opening is reversed. Very little power is required for the reversing of the engine by this means, and the engine-man can regulate the amount of expansion at will.

Counterbalancing.—If the winding-engine had to lift only one cage, it would have to be sufficiently powerful to lift the weight, not only of the coal, but of the waggons in which it is carried, of the cage, and of the whole length of the winding-rope down to the pit-bottom—a total weight say three to four times as great as the net load of coals. When there are two cages, one going up when the other comes down, they balance each other, and also the waggons balance, therefore the only load the engine has to lift is that of the coal and that of the rope. Just at the start the weight of the rope is very great, but when it is halfway the descending rope is as heavy as the ascending rope, and they balance; but when the load of coals is near the top, the descending rope may very likely overbalance the load of coals, and the winding-engine has no load at all to lift. In order, therefore, to equalize the load upon the engine, balance-weights are sometimes used. In the Monkwearmouth engine already described (see Figs. 499 and 511), heavy balance-weights, composed of two bunches of chain cable, each bunch weighing 5 tons, are suspended in a small pit over two flat pulleys. The bunches are attached to a flat chain; each chain is wound on a small drum on the main shaft of the engine, 3 feet 6 inches in diameter, rising to 8 feet 9 inches in diameter when the chain is wound up. When the engine starts to raise a load of coals, both these weights are hanging in the air, and materially assist the engine by balancing the weight of rope hanging down the shaft; as the engine proceeds and the rope is wound up, the balance-weights come to the bottom of the staple-pit, and by the time the engine has accomplished half its journey, the flat chain on the small drums is all unwound; as the engine goes on, the flat chain is necessarily wound up again; and before the coals arrive at the pit top the balance-weights are raised out of the staple-pit and are again hanging in the air, tending by their weight to stop the engines, and ready to help the engine to start on its next journey.

Another method of balancing more recently adopted at collieries (see Fig. 512) is to fasten a rope similar to the winding-rope to the bottom of each cage, the rope hanging down the shaft

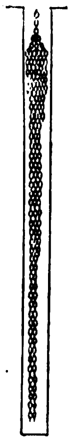


FIG. 511.—Balance-chain.

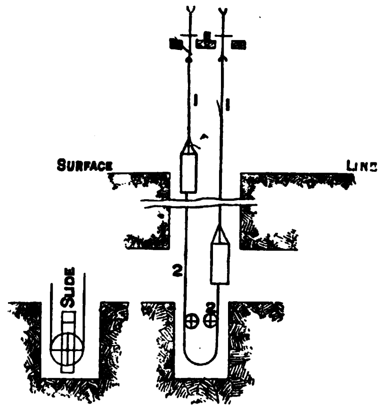


FIG. 512.—Balance-rope and pulley-guide. 1, winding-rope; 2, balance-rope.

under the top cage, going into the sump and coming up to the under side of the bottom cage; there is thus a perpetual balance-weight equal to the winding-rope. The balance-rope sometimes passes by two pulleys in the shaft-side, which guide it, and sometimes passes underneath a large pulley, which may move up and down in a vertical slide.

Scroll Drum.—Another method of balancing is by means of the scroll or spiral drum (see Figs. 502, 513). In this latter figure the engines are a pair of 36-inch cylinders, 6-feet stroke, and the drum is 20 feet on the smallest diameter, and 30 feet on the largest diameter. As the engine proceeds to wind up with this drum, the rope is wound up in spiral grooves on a continually increasing diameter of drum, until it finishes at a diameter half as large again as that at which it started; whilst the other rope,

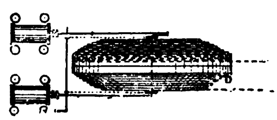


FIG. 513.—Spiral or scroll drum.

which is being lowered down the shaft, was on a large diameter at the start, and on a small diameter at the finish. The effect of this is that the rope which is being unwound goes at a speed of say three, whilst the rope which is being wound up at the start has a speed of say two, and consequently the descending waggons and rope have a leverage of three to two over the ascending cage, waggons, and rope, and in this way the weight of the rope is counterbalanced; and at the end of the journey, the loaded cage is wound up on a large diameter, whilst the descending rope and cage are on a small diameter. This helps to stop the engine, a thing as difficult as the starting.

Flat Ropes.—A certain amount of counterbalancing may be obtained in cases where the winding is done with a flat rope, especially if this rope is thick, as it would be if made of hemp or some other vegetable fibre, such as aloe. In Belgium it is very common to use ropes of aloe fibre 12 inches wide, and $1\frac{1}{8}$ inch in thickness. If the drum makes thirty revolutions, the diameter will be increased by thirty times $2\frac{1}{4}$ inches = $67\frac{1}{2}$ inches; thus, if at the start the diameter of the rope-roll was 5 feet 6 inches, at the finish it would be 11 feet. When the cage and empty waggons descend, they are suspended from a rope-roll at its largest diameter, and counterbalance the weight of the rope that is ascending (see Fig. 514).

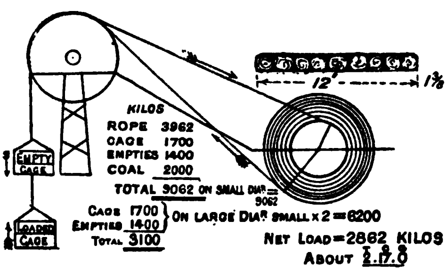


FIG. 514.—Winding with aloe fibre in Belgium.

Shaft Indicators.—The engine-man must have before his eyes a dial-plate or other contrivance to indicate the position of the cage in the shaft. This is sometimes done by attaching a small cord to a shaft connected to the engine-shaft by spur-gearing; this cord passes over a pulley and has a weight suspended at the end; this weight is raised up and down in a slide; marks on the slide correspond with the bottom and top of the shaft on a reduced scale. When the weight is near the top, it strikes against the lever of a bell, and so warns the engine-man to be on the look-out; and when the weight is near the bottom of the slide, it strikes against another bell, or sometimes the cord is so attached to the drum that it is all unwound at the middle of the journey, and wound up again towards the end of the journey. In this case the weight is always at the top of the slide when either cage is at the top of the shaft. Another kind is to fix opposite the engine-man a dial-plate like the face of a clock, and by means of bevelled gearing, connected with the main shaft of the engine, a pointer is made to revolve, and show on the dial face the position of the cage, and when it is near the top a bell is struck.

Regulator.—Every winding-engine is fitted with a regulator or throttle-valve, by which the engine-man can shut off steam altogether, or let it on in some small proportion. Sometimes this valve, which is usually a double-beat valve, is prevented from entirely closing, so that it may be more easily opened wide by the engine-man.

Brake.—Most winding-engines are fitted with a foot-brake, that is to say, a brake on the drum, the lever operating which is under the engine-man's foot, so that, whilst both his hands are engaged, one with the regulator valve and the other with the reversing lever, he may be able to apply the brake with his foot.

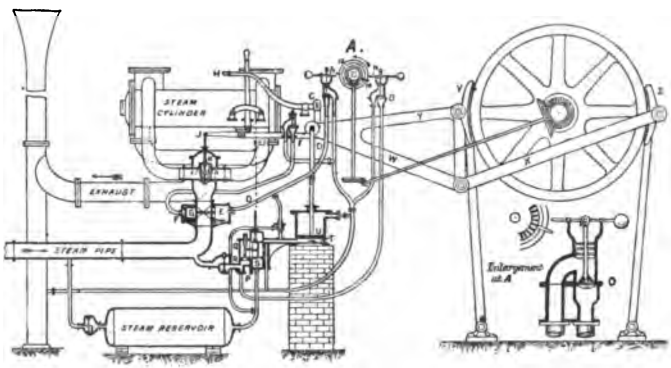


FIG. 515.—Automatic steam-brake and shut-off (M. E. Reumaux).

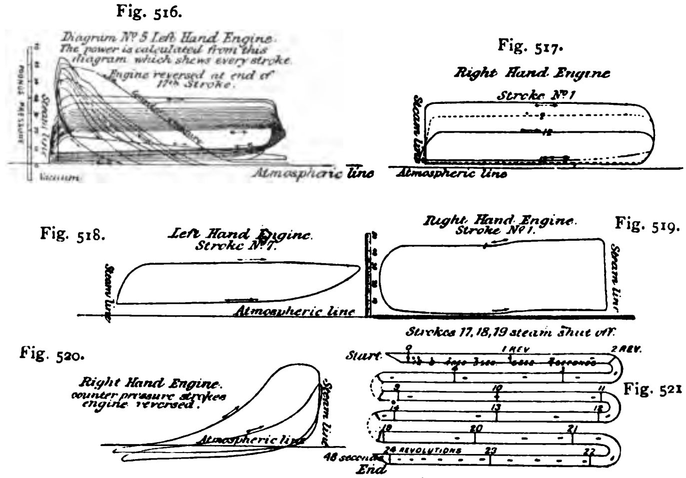
Most large winding-engines also have a steam-brake, which is similar to the foot-brake, except that the pressure is applied by steam. The steam-brake is sometimes applied by hand, and sometimes by foot. The same brake-strap is often both foot and steam brake, the foot lever and the steam-engine lever being both connected, so that the foot can apply the brake first, the steam being only used occasionally. A powerful steam-brake is shown in Fig. 515, described further on.

Automatic Overwind Prevention.—Some winding-engines are fitted with apparatus to stop the engine in case the engine-man should forget, or be unable, to stop it at the right place. Some of these are designed on the principle that the cage, when lifted up above the pit-bank, shall come in contact with a lever, a rod from which is connected with the steam-valve and the steam-brake, and first shuts off steam and then applies the brake.

A very good apparatus to prevent overwinding has been designed by Monsieur E. Reumaux, the chief engineer of the large collieries at Lens, in the north of France. This design is shown in Fig. 515. At A is a valve-wheel, which is driven by means of bevelled gearing from the main shaft of the winding-engine; attached to this wheel are tappets. When the engine has gone far enough, it brings the tappet *b* against the lever of the small valve C, thus opening the valve and allowing the steam in the small pipe D D to escape into the exhaust main, and thus allows the steam in the cylinder E to escape; and the steam in the other end of the cylinder at F then forces the piston G across the steam-pipe, and so closes it, shutting steam off from the engine. If the engine-man has already anticipated this automatic steam-valve by lowering the handle H, and so opening the small valve I, the steam from F will have been allowed to escape, and thus the automatic valve G will not move. Thus this valve only moves in case the engine-man shall have forgotten to shut off steam by the regulator, because by lowering the handle H the lever J is depressed, and the throttle-valve or regulator K is closed. If the engine is not immediately stopped by this closing of the steam-pipe, the tappet M comes in contact with the lever N, opening the small valve O, and so permitting the escape of the steam from the smaller cylinder P. The upper part of this cylinder, Q, is in connection with the steam-pipe, and the small piston R is thus forced down, bringing with it the valve S, and so opening the port into the brake cylinder T, thus lifting the piston U and raising the levers V and W and tightening the connecting bar X, by which means the brake-blocks Y and Z are drawn tightly together against the brake wheel. The brake can be taken off by the engine-man lifting the slide-valve S by means of his hand-lever.

Drop Pits.—In some mines, particularly in French thick-coal collieries, a good deal of stone has to be lowered down the pit. This is sometimes done by means of a single cage and balance-weight, the pulley or drum being stopped by means of a brake.

Signals.—The engine-man is guided by signals which ring into the engine-room from the banksman, and also from the pit-bottom. The bells which give the signals are often worked by means of levers, wire-strand, and bell-cranks. Electric bells, however, are often used with a great economy of time. In some metal mines there are a great many landings between the top and the bottom, at which the overlookers and workmen have to get out, and it is convenient to be able to signal to the bank and engine-room from the cage. An ingenious apparatus has been used at the Himmel's Furst Mine, near Freiberg. The galvanic battery is in the engine-house; one pole is connected with the engine-cylinder, and through that with the drum and winding-ropes; the other pole is connected with a copper strip, which is fastened to one of the wooden conductors all the way down the shaft; thus if contact is made between the cage and this copper strip, there is a complete circuit. In order to make a signal, a slide on the cage is pressed by an eccentric against the copper strip; then the signal "key" is pressed against a knob on the



FIGS. 516-521.—Indicator diagrams.

cage, thereby making contact. The number of times the key is pressed represents an equal number of signals sent to the bank.

Power.—It is important for a colliery manager to know the power required by a winding-engine. The most practical way of obtaining this information is by examining the work done by existing engines. This has been done in the case of the winding-engine shown in Fig. 504 (and already described), by means of an indicator (see Figs. 299, 300, 301). The diagrams shown in Figs. 516, 517, 518, 519, and 520, were taken from the engines when at work.

Diagrams in this case were taken from both ends of both cylinders, which, of course, should always be done. It is advisable to have separate indicators at each end of each cylinder, so that the pipe leading from the cylinder to the indicator should be as short as possible. This is better than the practice sometimes adopted of having a "breeches" pipe connecting both ends of the cylinder, in the centre of which the indicator is placed. Diagram Fig. 516 shows every revolution of the engine from the start to the finish of winding up a load from the pit-bottom to the pit-top, the number of revolutions being $23\frac{3}{4}$. It will be seen that no two strokes show the same steam-pressure nor the same back pressure. This is due to the fact that no two strokes are made at the same speed during acceleration, and as the speed increases, the steam is unable to follow up the piston, partly owing to the friction in the steam-pipes, and partly owing to the want of time, owing to the steam-valve not opening soon enough, as already described. And, in the same way, the back-pressure line rises every stroke during acceleration, because the steam is unable to get out of the cylinder sufficiently quickly. This would be, to a great extent, remedied by giving the steam and exhaust valve a little "lead." It will be observed that steam is cut off at the beginning of the seventeenth revolution, and that no more steam is given to accelerate the engine after the conclusion of the seventeenth revolution. The engine has now attained its maximum speed, and it is necessary to endeavour to stop it. This is done by reversing the engine, throwing the steam-pressure from the boilers against the advancing piston, which causes the back-pressure lines to rise up, crossing the steam-pressure lines as shown in the strokes marked counter-pressure, which represent the pressure against the piston. These counter-pressure strokes do not represent any corresponding consumption of steam, as the steam in the cylinder is simply forced back again into the steam-pipes. Two counter-pressure strokes are shown separately in Fig. 520. The working of the valves and the effect of speed is shown in Figs. 517, 518, 519. On referring to the scale, it will be seen that the pressure in the cylinder at the beginning is about 43 lbs., whereas when the engine has attained full speed it is only about 23 lbs., and at the same time the back pressure in the cylinder on the return stroke at full speed does not fall below 7 lbs., whilst the average back pressure for that stroke is a great deal more.

The speed of the engine was measured by an instrument which the writer constructed, called a velocimeter (see Figs. 522 and 300). By this apparatus, a strip of paper is drawn under a pen attached to a lever, so that a mark is made every second on the paper. The strip is drawn by the engine, and the faster it goes the greater the distance the marks are apart. This is shown on Figs. 300, 521, and 523 on a reduced scale. On referring to Fig. 521, where the strip of paper is shown coiled up, it will be seen that during the first revolution the pen has made six marks, showing that 6 seconds were occupied in that revolution; during the second revolution the pen made two marks, and the paper was drawn halfway towards the third mark, showing that $2\frac{1}{2}$ seconds were occupied during that revolution; during the third revolution the pen also made two marks, and it will be seen that about $1\frac{7}{8}$ second were occupied in that revolution; during the fourth revolution the pen also made two marks, and it will be seen that the time occupied was about $1\frac{3}{4}$ second; during the fourteenth revolution the pen only made one mark, and the time occupied was very little more than 1 second; during the twenty-fourth revolution, when the engine was being brought to a stop, the pen made seven marks, showing that 7 seconds were occupied in that revolution.

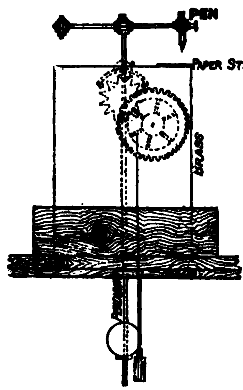


FIG. 522.—The velocimeter.

By means of the indicator-diagrams the steam-power employed was calculated and was drawn on the explanatory diagram (Fig. 523), and shown by the line on which the following words are written: "Line showing indicated steam-pounds for each revolution of the engine." The horse-power of the engine is also shown on another line. This was obtained with the aid of the velocimeter diagram, showing the time in seconds for each revolution. By means of the speed of the piston calculated to feet per minute, and the steam-pressure, and cylinder area in square inches, the horse-power is calculated and marked in the figure; the dead weight to be lifted is ascertained, and the work done in lifting every revolution is found by multiplying the dead weight by the distance travelled, the distance being in this case (the drum being plain for round rope) the same every revolution, though the weight varies every revolution, because one rope is being wound up as the other comes down. The work done in lifting the dead weight is shown by the line with the following words: "Line showing power in foot-pounds required to balance the weight of coal and rope." It will be seen that this power is about 600,000 foot-pounds the first revolution, and that by the end of the journey the weight of the descending rope actually overbalances the coal to be lifted. In addition to the weight to be lifted, there is the friction, the amount of which is shown on the diagram. But the work done in the first revolution is 500,000 foot-pounds more than that required to lift the load and overcome the friction; this extra power is expended in giving velocity. The power theoretically required for acceleration was also calculated

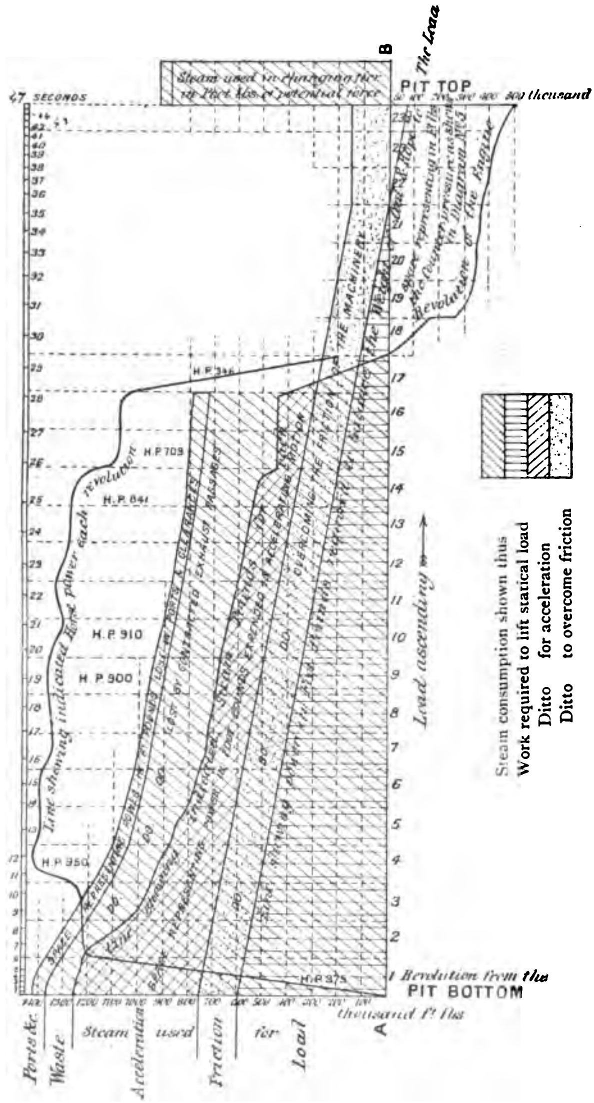


FIG. 523.—Winding-engine (Denaby). Explanatory diagram. Each dotted rectangle contains 100,000 foot-pounds.

from the weights of the moving parts and their velocities as measured. It will be seen from the diagram that a great deal of power is required to give velocity, that the engine has to be reversed to stop it when it has made sixteen revolutions, and that a great deal of back pressure is required to bring the machinery to a stop. It will also be seen that a great deal of power was lost in driving the steam out of the cylinder. This might be obviated, partly by altering the adjustment of the valves and partly by adopting some system of expansion, and very likely this may have been remedied since these diagrams were made.

Fig. 524 shows a similar diagram for the Monkwearmouth engine, already described in pp. 392 and 404, and Figs. 499 and 511, but the line representing the power required to lift the weight of coal and ropes, instead of being a straight line, is very much bent. This is due to the action of the balance-chains which facilitate the starting and stopping of the engine. (If there were a third balance-weight equal to one of the other two, the dotted line represents the power that would then be required to lift the weight of coal and rope.)

It will be seen that at the start the power required to lift the load is equal to about 500,000 foot-pounds per revolution, while at the end it is about 260,000 foot-pounds per revolution. Following out the action of the balance-weights, it will be seen that the load-line quickly falls during the first five revolutions, owing to the winding up of the rope with the loaded cage and the descent of the rope with the empty cage. By the end, however, of the fifth revolution, the balance-weights have begun to rest at the bottom of the staple-pits, and by the beginning of the eighth revolution both bunches of chain cable are resting on the ground; thus their weight no longer assists the engine. As the load-line has risen up to as much as it was at the start, again it quickly descends, owing to the winding up and unwinding of the ropes, till at the beginning of the twelfth revolution the engine begins to pick up the flat chain, which causes a slight increase of the load. Again, however, the load-line falls by the winding and unwinding of the ropes until the end of the sixteenth revolution, when the bunches of cable are partly lifted, and as they are raised from the bottom the load against the engine increases till the beginning of the nineteenth revolution, when both balance-weights are suspended, and again the load-line begins to fall. It must be noticed that, in a great measure owing to the balance-weight, less counter-pressure is required in this case to stop the engine than was necessary with the Denaby engine.

Fig. 525 shows a similar diagram for the Douglas Bank Colliery. In this case a spiral drum is used, giving a more even load-line than in the case of the other two. This engine was a coupled horizontal engine, two cylinders, each 30 inches diameter and 5-feet stroke; maximium pressure in cylinders, 49 lbs.; maximum

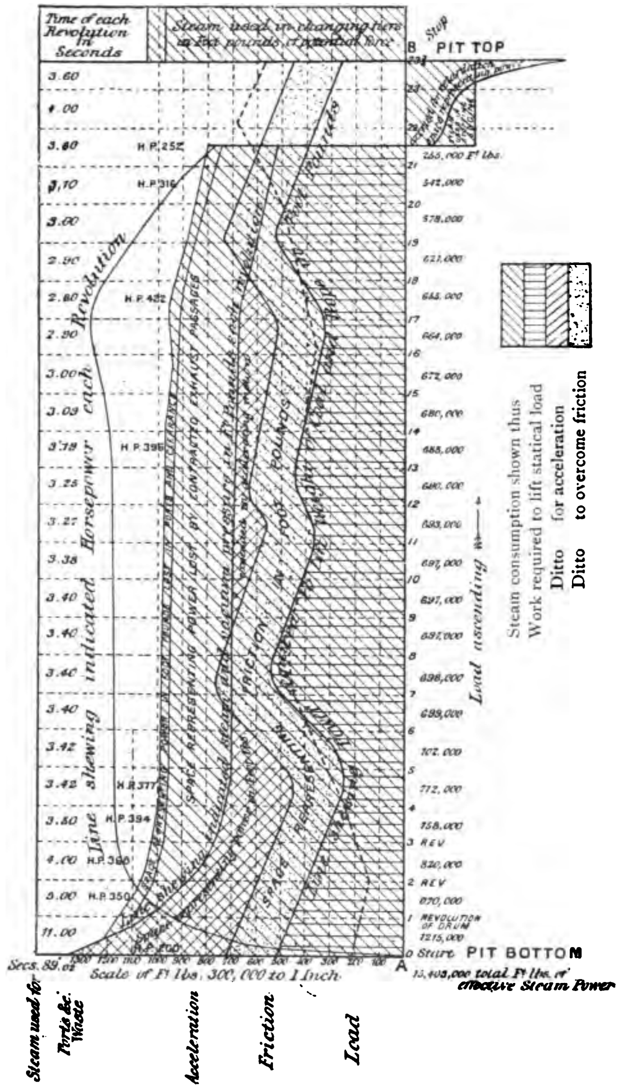


Fig 524.—The Wilson winding-engine, Monkwearmouth Colliery. Explanatory diagram. Each dotted rectangle contains 100,000 foot-pounds.

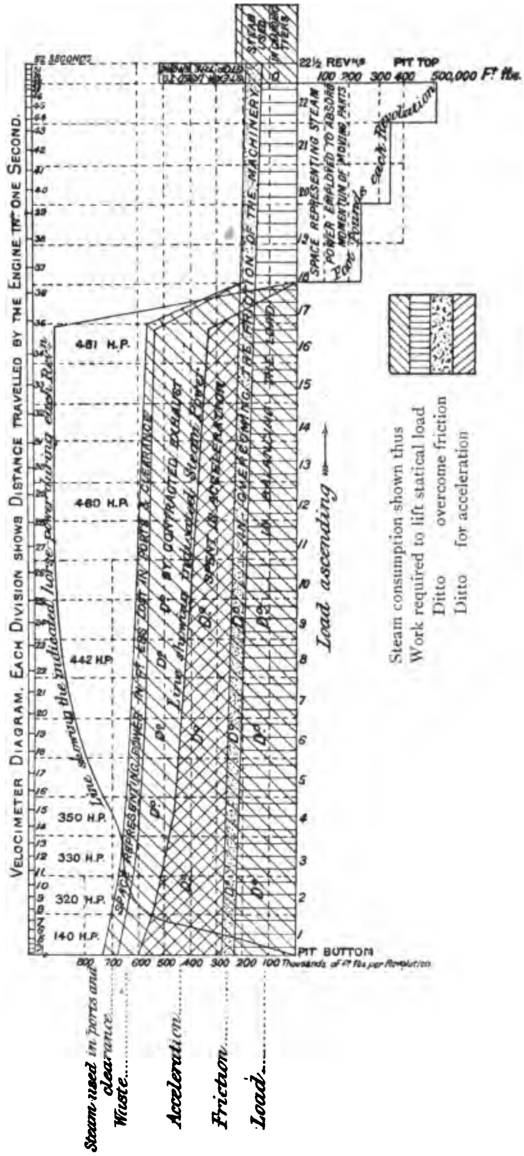


FIG. 525.—Winding-engine, Douglas Bank Colliery. Explanatory diagram. Each dotted rectangle contains 100,000 foot-pounds.

velocity of piston, 462 feet a minute; mean velocity, 260 feet a minute; the minimum diameter of the drum, 18 feet, and the maximum diameter, 25 feet 4 inches; weight of drum about 82,000 lbs.; total weight of moving parts, including engine, drum, ropes, pulleys, chains, and coals, 133,900 lbs.; steel-wire ropes, 11 lbs. per fathom; cage, 2240 lbs. of steel, containing 4 tubs, each weighing 336 lbs. and holding 6 cwt. of coal. Depth of pit, 1530 feet; time of winding, 52 seconds.

It will be observed that the larger diameter is only about 29 per cent. greater than the smaller diameter. In the case, however, of the engine at Harris's Navigation (Figs. 502 and 503), the maximum diameter of the drum is 32 feet, and the minimum diameter is 18 feet, or nearly 44 per cent. less than the maximum diameter. The consequence of this increased difference is a more effective balancing, the power required to lift the load of coal and ropes being nearly even throughout the journey.

This is proved by the following calculations, which show the work to be done by the engine in lifting the dead load during the first revolution, the middle revolution, and the twenty-fourth revolution.

The work done in lifting the dead load is calculated in foot-tons as follows:—

FIRST REVOLUTION.

	Tons.	Average circumference of drum.			Work done in lifting the dead load on the engine in foot-tons per revolution.
Loaded cage and rope	16·0 ×	18·5 ×	3·14 =	930	
Empty cage and rope	4·75 ×	32·0 ×	3·14 =	476	454
				454	

MIDDLE REVOLUTION.

	Tons.	Average circumference of drum.			
Loaded cage and rope	13·75 ×	30·5 ×	3·14 =	1317	
Empty cage and rope	7·75 ×	29·5 ×	3·14 =	720	597
				597	

TWENTY-FOURTH REVOLUTION.

	Tons.	Average circumference of drum.			
Loaded cage and rope	10·75 ×	32 ×	3·14 =	1079	
Empty cage and rope	10·0 ×	18·5 ×	3·14 =	581	498
				498	

The above weights are made up as follows:—

	First Revolution.				Middle Revolution.			
	Loaded cage.		Empty cage.		Loaded cage.		Empty cage.	
	tons.	cwt.	tons.	cwt.	tons.	cwt.	tons.	cwt.
Cage and chains ...	2	10	2	10	2	10	2	10
Tubs	2	0	2	0	2	0	2	0
Coal	6	0	0	0	6	0	0	0
Rope	5	10	0	5	3	5	3	5
	16	0	4	15	13	15	7	15

Twenty-fourth Revolution.

	Loaded cage.		Empty cage.	
	tons.	cwt.	tons.	cwt.
Cage and chains	2	10	2	10
Tubs	2	0	2	0
Coal	6	0	0	0
Rope	0	5	5	10
	10	15	10	0

It is evident, from the above calculations, any alteration in the section of the rope, or weight of cage, tubs, and coal lifted, will materially affect the dead load on the engine.

Engine Power.—From these diagrams it may be seen that the total power required by an engine is from two and a half to three times that which would be necessary merely to balance the dead weight, and that at least as much power is expended at the start in giving velocity as in lifting the load. This is not only because of the high speed attained by the cages in the shaft, which in some cases reaches a mile a minute, but because of the short time allowed for attaining such a high speed. Engines are often going at a great velocity within ten seconds of starting, and within thirty seconds have reached nearly their full speed, though this speed continues to increase until the steam is shut off.

Lubrication.—It is important that all the parts of a winding-engine should be well lubricated to reduce the friction. Thirty years ago tallow was almost universally used for lubricating the valves and piston. It has, however, been found that tallow has an injurious effect when used inside a steam-cylinder, corroding the metal. Mineral oil or grease is now generally used, some heavy oils being specially made suitable for the interior of valve-chests and cylinders. Mineral oil is also chiefly used for the slides and bearings. Metallic packing has also come into use for the stuffing-boxes, and some engineers prefer it to all kinds of soft packing.

CHAPTER XIX.

PIT-FRAMES, PULLEYS, CAGES, WATER-TANKS, CONDUCTORS, HYDRAULIC STAGES, DOUBLE STAGES, ROPES, ETC.

THE pulleys for a winding-rope are generally of large diameter, sometimes reaching 20 feet for a steel rope about 1½ or 1¾ inch in diameter. The diameter of a pulley ought to be in some way proportional to the diameter of the wires of which the rope is made, and also in some way proportional to the diameter of the rope. A rope may work round a pulley only 3 feet in diameter, but it will endure longer the greater the diameter of the pulley. For a winding-rope only 1 inch in diameter, a pulley 10 feet in diameter may give satisfactory results; but for large collieries, with steel ropes 1½ to 1¾ inch in diameter, the pulley is generally from 14 to 16 feet. It is generally made with a cast-iron grooved rim (see Fig. 526), with wrought-iron arms cast in, and also cast into a central cast-iron boss. The boss is cast in two halves, split across the diameter, so that the pulley-rim may freely contract when cooling. The two halves of the boss are afterwards united by wrought-iron collars shrunk on, and then bored and key-gated. Flat rope pulleys are constructed in a similar manner, merely altering the shape of the groove. Sometimes the rim is made of wrought-iron or steel plates riveted together, with wrought-iron or steel arms riveted on (see Fig. 527, of a 20-foot pulley for Harris's Navigation).

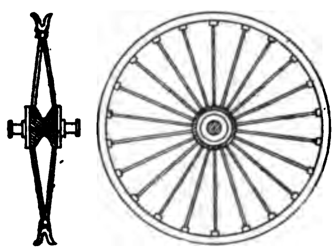


FIG. 526.—Cast-iron pulley.

The details of this pulley are given in Fig. 527. The shaft is of wrought iron, 10 inches in diameter, and 4 feet 10 inches long over all. The journals are about 8½ inches in diameter, and 12 inches long. The boss is of cast iron, about 3 feet 6 inches in diameter, keyed on to the shaft, which is turned to fit. The rim is carried by forty-eight round steel arms, $1\frac{5}{16}$ inch in diameter. These arms are flattened at the end nearest the rim, to which they are attached by small steel plates riveted on to the arms and

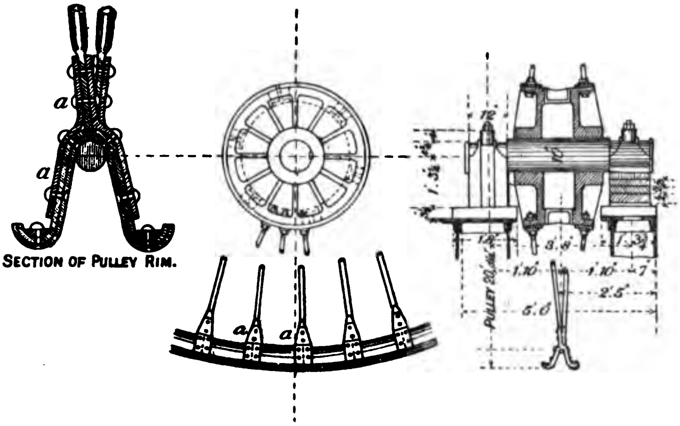


FIG. 527.—Steel pulley (Harris's Navigation).

also on to the rim, as shown in the figure at *a*. The rim is made of two layers of curved steel plates, breaking joint with one another, and riveted together. The ends of the arms at the boss are screwed and fixed by nuts on both sides of the boss flange. The arms can thus be tightened to the extent requisite to bring the rim into a true circle. The curve of the rim is of just sufficient size to take the rope at the bottom, which is about $1\frac{9}{10}$ inch in diameter. The pedestals have under brasses, but no top brasses. Beneath the under brasses are three blocks of indiarubber, each two inches thick, separated by steel plates. These are intended to act as a spring to diminish the strains on the rope.

Some pulleys 20 feet in diameter have been made with cast-iron rims, turned in the lathe to secure perfect smoothness and truth in running.

A rule sometimes given for the diameter of pulleys is eight hundred times the diameter of the wires of which the rope is made. This rule may suffice for haulage and transmission of power, but gives an insufficient diameter for winding-rope pulleys. For haulage ropes of steel 1 inch diameter, pulleys 5 to 7 feet are commonly used with good results.

Pit-frames.—The pulleys are carried on a frame of timber or iron, so that the centre of the pulley is from 20 to 80 feet above the level of the bank. The height of the pulley above the bank depends, first, on the height of the cage (thus a four-decked cage may be 20 feet in height); next, on the length of the chains by which the cage is suspended from the rope; third, on the safety-hook and catch-plate, requiring, with the cage-chains, another height of 12 or 14 feet. There should also be room to raise the cage above the bank whilst the other cage is being lowered into the sump, say 6 feet; adding half the diameter of the (say 16-feet) pulley, 8 feet, gives a total minimum height from bank to centre of pulley, 48 feet. The height of pit-frames at large collieries is often from 60 to 80 feet above the bank. Wrought-iron pulley-frames are sometimes in the shape of a plate girder (see Fig.

FIG. 528.—Pit-frame, plate girder (Hasard).

528), and sometimes of a lattice girder (see Fig. 529), sometimes a box-girder and lattice-girder frame combined (see Fig. 530), and sometimes the upright and backstays are cylindrical plate-iron riveted columns, similar to ships' masts. Examples of wooden pit-frames are shown in Figs. 531, 531*a*.

Rope-socket.—The attachment of the winding-rope to the cage may be done in several ways. The end of the rope is sometimes, in the case of flat ropes, turned round a grooved link and clamped, as shown in Fig. 532. More often the end of the rope is put into a socket made of wrought iron (see Fig. 533), which is larger at the bottom than the top. The socket is generally split, and has a loop at the bottom through which a shackle can be placed connecting it to the cage-chains; about 4 feet of the rope-end is untwisted, and the part above is bound tightly round with copper wire. The ends of the wires are now turned back, some of the wires for the whole 4 feet, others are cut off rather shorter, and in this way a lump is formed at the end of the rope where the wires are bent round. The wires so bent back are then closely bound round with wire; this is placed in the socket; over this wrought-iron rings are tightly hammered down, closing the two sides of the socket, so that the rope cannot be drawn through. Sometimes the two sides of the socket are closed by rivets; a pricker is put through

FIG. 529.—Pit-frame, lattice girder (Mansfeld).

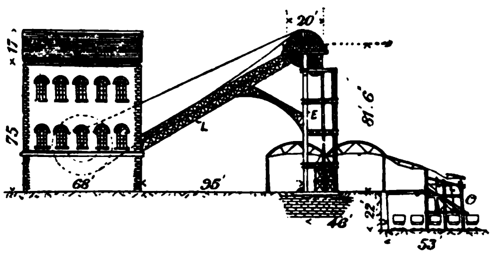


FIG. 530.—Pit-frame, lattice girder (Harris's Navigation).
E, box girder; L, lattice girder; C, screens.

the rope to open a way for the rivet. A solid socket is sometimes used, in which the rope jams itself. A rope is more likely

FIG. 531.—Wood pit-frame.

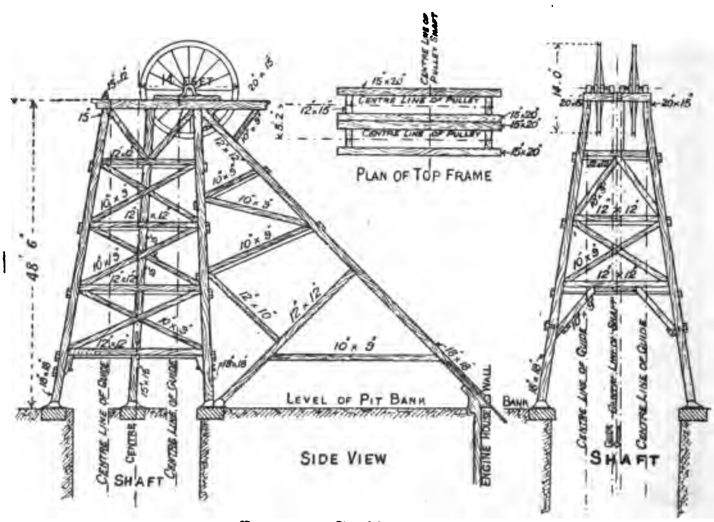


FIG 531a.—Double pit-frame.

to draw through the socket than to break, and, unless the socket is well fastened on, the rope is very likely to draw out. When the wires of the rope-end are bent back, the part of the rope above is often not bound with wire, but the ends of the wires

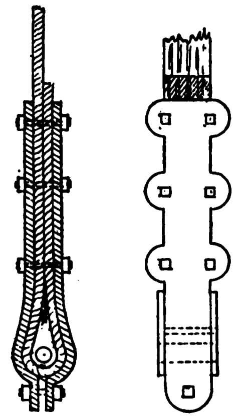


FIG. 532.—Rope-clamp for flat rope.

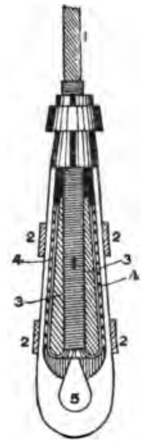


FIG. 533. — Round rope-socket and capping. 1, rope; 2, collars; 3, cone round which wires are turned back; 4, wires turned back over cone; 5, for cage attachment.

are threaded into the rope, so that they are fast. Another method is to slip an iron cone over the rope-end; the wires are bent back over this cone, and then bound with wire; this is then placed in a conical socket, as in the case above described. Sometimes the ends of the wires are bent back forming a cone, which is drawn into a conical iron socket; melted white metal is then poured into the socket, which firmly fixes the wires. This plan is adopted by Professor Goodman when testing ropes at the Yorkshire College.

It must be understood that the capping illustrated in Fig. 533 is only a diagram, and not a working drawing. For a load of say 10 tons of cage and contents, a very strong cap is required. If each wire is hooked at the end and secured in lead hardened with antimony, a short cap, sufficiently wide to hold all the wires and the lead, will suffice. If the lead mixture is not used, a long cap grasping the rope for a length of at least 3 feet is necessary, so as to avoid all risk of the wires being broken one by one; six collars should be driven over the split cap, of sufficient strength to withstand the wedge-like action of the load.

The caps are generally cut off at intervals varying between six weeks and six months, and with them 6 to 12 feet of rope, if that length can be spared, and thus a fresh part of the rope is brought to bear on the pulley when the cage is at the top.

In the following table are given the results of some tests made for the author at the Yorkshire College, on specimens supplied by

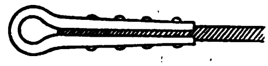
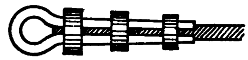
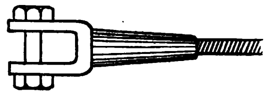
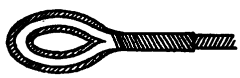
	Breaking strain of rope as given by makers.	Maximum load.	Remarks.
	Tons. 30	Tons. 14·9	Capel with rivets. Top rivet of capping sheared. Rope pulled out from cap.
	30	8·5	Capel with collars. Capping failed. Rope pulled out from cap.
	30	10·5	Conical socket. Wires bent back. Capping failed. Rope pulled out from cap.
	30	26·7	Eye, spliced in. Capping failed. Splice gave way.

FIG. 533*a*.—Tests of rope caps.

him. These show the relative strengths of different kinds of rope capping. In the four cases given, the makers of the ropes capped them, and the breaking strength of the rope was the same in each case; a perfectly efficient cap should of course be as strong as the rope, but it will be seen that this is rarely the case.

Quality of Ropes.—Fifty years ago hemp ropes were generally used; they were superseded by iron-wire ropes. About twenty-five years ago steel ropes were introduced; these are used almost universally (in England), to the exclusion of any other kind, and the strongest kind of steel, called in the trade "plough steel," is now frequently employed. The advantage of the best steel is that a light rope suffices, thus reducing the strain upon the engine and upon the brake; and steel endures much longer than iron, thus there is greater economy. The strength of steel ropes is given in the lists of various makers, from which the opposite table is extracted.

The following rules give roughly approximate results, and will be easily remembered. (1) To find the weight of a round wire rope made in the ordinary way with twisted strands: The circumference in inches squared = weight per fathom in pounds. (2) To find the breaking strain: The weight per fathom in pounds multiplied by 2 = breaking strain in tons for Bessemer steel.

TABLE XXII.—TABLE OF ROUND WIRE ROPES.

Circumference in inches.	Weight per fathom in lbs.	Bessemer steel.			Patent crucible steel.			Improved plough steel.		
		Breaking strain in tons.	Working load in cwts.		Breaking strain in tons.	Working load in cwts.		Breaking strain in tons.	Working load in cwts.	
			Shafts.	Inclines.		Shafts.	Inclines.		Shafts.	Inclines.
1⅛	1·3	2·15	4·75	6·00	3·50	8·00	9·75	5·33	11·66	15·00
1¼	1·75	2·75	7·00	9·00	4·37	10·00	13·75	6·25	14·50	18·00
1⅜	2·12	3·50	8·00	11·00	6·50	16·00	21·00	8·00	18·75	22·25
1½	2·25	4·37	10·00	13·75	7·35	17·00	24·00	8·66	19·66	24·66
1¾	3·12	5·50	12·00	17·33	8·83	19·66	28·33	13·00	29·00	41·66
2	4·10	7·35	17·00	24·00	11·84	26·41	38·00	17·33	37·83	54·66
2¼	4·91	8·83	19·16	28·00	15·50	33·75	48·33	21·66	47·83	69·33
2½	6·17	11·17	24·75	35·66	19·00	42·00	60·66	26·50	57·92	83·66
2¾	7·58	13·33	29·66	42·66	23·00	50·33	72·33	32·00	70·33	102·33
3	8·58	15·50	33·75	48·33	27·17	59·58	86·00	37·00	81·66	117·66
3¼	10·10	19·00	42·00	59·50	30·30	67·25	96·33	43·33	96·33	139·00
3½	12·10	22·00	48·83	70·33	36·00	79·00	112·00	51·33	115·33	166·00
3¾	14·00	24·66	56·00	80·00	40·66	89·66	129·33	57·50	131·00	184·50
4	15·87	27·25	64·38	90·50	49·50	113·00	154·50	66·00	155·50	219·50
4¼	17·50	30·50	72·37	101·50	55·00	127·00	183·00	75·50	178·00	251·50
4½	20·00	37·50	79·00	110·00	63·00	145·00	193·00	87·50	204·00	288·00
4¾	21·83	42·00	91·00	125·00	70·00	155·00	218·00	99·50	221·00	305·00
5	23·80	47·20	105·50	145·70	79·50	170·50	245·00	109·50	244·00	336·50
5¼	25·50	52·00	117·00	161·50	87·50	195·00	268·70	119·00	271·50	364·40
5½	27·50	55·50	123·50	170·20	94·50	210·00	290·00	131·50	294·50	406·50
5¾	30·50	60·50	134·50	185·75	103·00	229·00	316·40	144·00	320·00	441·50
6	34·00	66·00	147·00	202·85	111·00	247·00	340·75	156·00	346·50	478·65

Multiplied by 3 for "patent crucible steel," and by 4 for "improved plough steel."

Example.—Let the wire rope be 4 inches circumference: $4^2 = 16$; therefore the weight of the rope is about 16 lbs. per fathom. In the table it is given as 15·87 lbs. per fathom.

The breaking strain for Bessemer steel		$= 16 \times 2$
		= 32 tons.
" " "	"patent crucible steel"	$= 16 \times 3$
		= 48 tons.
" " "	"improved plough steel"	$= 16 \times 4$
		= 64 tons.

On referring to the table, it will be seen that the breaking strains are 27·25 tons, 49·50 tons, and 66 tons respectively.

In winding-ropes there should be a large factor of safety, in order to avoid the danger of breakage; that is to say, the rope should be very much stronger than the supposed working strain. Thus if the load upon the top end of the rope, when the cage is fully loaded and the rope is all down the shaft to the bottom, is 10 tons, the rope should be capable of bearing a strain of 100 tons before it will break, the factor of safety being 10.

Steel ropes, as now used, are generally made with a hemp core (see Fig. 534), and sometimes with a wire strand for the core. If an ordinary round rope is suspended over a pulley, and a weight hung at the end, the effect of the weight is to cause the rope to untwist. When this weight is a cage fastened to conductors, the rope is prevented from untwisting to any great extent, although as soon as the rope gets slack, as when the cage rests on the props, it curls, twisting the chains, which uncurl when the weight comes on. Ropes have been made that will not untwist with a loose weight, such as a sinking-hoppet; this is achieved by twisting the strands in a different way.

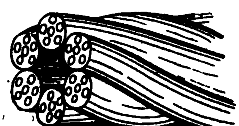


FIG. 534.—Wire rope.

There is a new kind of rope, called the "locked coil rope," patented by Messrs. Latch and Batchelor (see Fig. 535).[1] This does not untwist when loaded, and can therefore be used for sinking pits.

The following is a description of a rope made in 1880 for Harris's Navigation, described by Messrs. T. Forster Brown and G. F. Adams. The diameter was about 1·9 inch. It was made of the best selected steel; the gauge of the wire No. 11 B.W.G.,

[1] Their names are not now associated with this rope, as the patent is worked by others.

19 wires to each strand; six strands formed the rope, or 114 wires in all. Each strand was made by plaiting six wires round one wire, and then other twelve wires were plaited round the first group in the reverse way. The six strands were plaited round a hempen core. The weight of the rope below the pulley-sheave (700 yards) was about 5 tons. The calculated breaking strain was about 104 tons. The load it had to lift was as follows: Two trams of coal, 3 tons; two trams, 1 ton; cage and bridle, 2 tons 5 cwt.; rope, 5 tons; total, $11\frac{1}{4}$ tons. It was intended at a future date to increase the weight of coal, trams, cage, and bridle to $10\frac{1}{2}$ tons, and it was then intended to use a rope of the same size

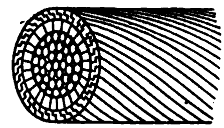


FIG. 535.—Rope: Latch and Bachelor's locked coil.

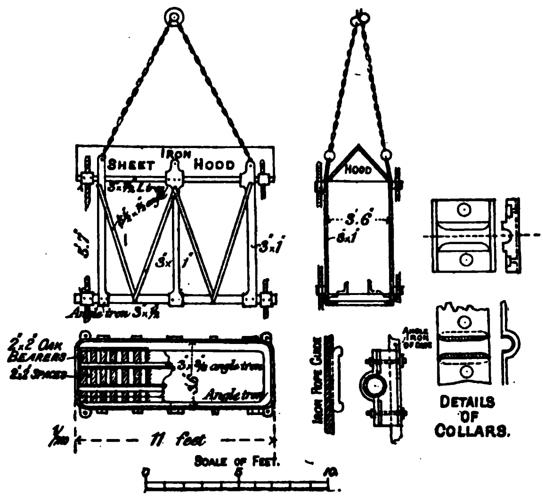


FIG. 536.—Cage and conductors.

as that above described, but to be made of the best plough steel. It was also intended to adopt a finer gauge of wire when the grooves of the drum were worn smooth.

Cages.—Cages have one, two, three, and four decks (and even perhaps more). Some examples of iron cages are shown in Figs. 536, 536*a*. Fig. 536 shows a single-decked iron cage, intended to hold two corves, each corf holding about 10 cwt. of coal. The cage is guided by four iron-wire conductors, and four cast-iron collars are bolted on to the top and bottom frames of the cage, making eight collars in all, the details of which are shown in the figure. Fig. 536*a* shows a double-decked iron cage, each deck to hold two corves, holding 10 cwt. of coal each, or three corves holding 7 or 8 cwt. each. The weight is about 2 tons. It is guided in the same way as in Fig. 536, there being eight cast-iron collars on the cage. The iron hood at the top of the cages is to shelter the men from falling fragments.

Cages are now generally made of steel. One is shown in

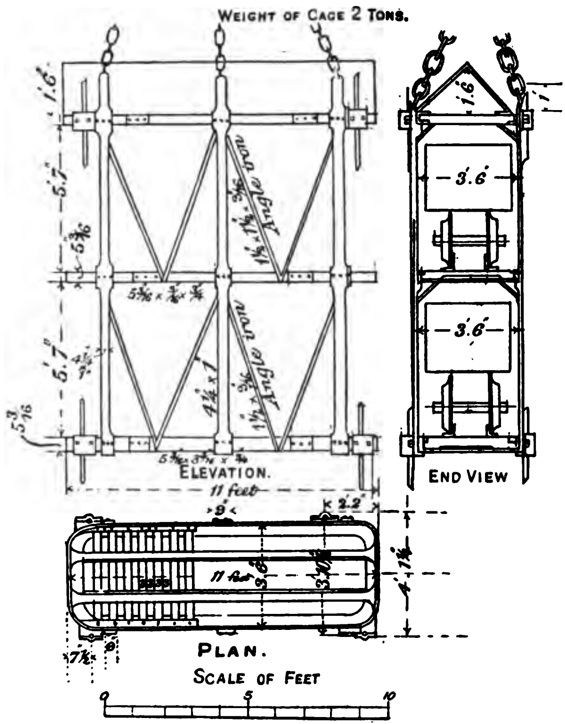


FIG. 536*a*.—Double-decked cage with three trams on a deck.

Fig. 537. This is reduced from one in the "Proceedings of the Institution of Civil Engineers," drawn by Messrs. T. Forster Brown and G. F. Adams. It is 12 feet long, and 4 feet 2 inches wide; will carry two trams on one deck, the height over all being 7 feet. The hood or covering of the cage is a flat steel plate. The weight, without bridle chains, is 33 cwt. There are six chains (called bridle chains) to attach the cage to the shackle. The trams raised in this cage are of iron, with steel channel sills and cross-stays.

They are 5 feet $10\frac{1}{2}$ inches long over buffers, 3 feet 4 inches wide, and 3 feet 5 inches above the rails. The wheels and axles are of steel, 15 inches and $1\frac{3}{4}$ inches in diameter respectively, the gauge of the rails being 2 feet 6 inches. Each tram holds 30 cwt. These were the first cages adopted at Harris's Navigation, it being intended to substitute double-decked cages at a later date. Cages are now sometimes made capable of taking four corves on a deck; in one case five corves, and in another case six corves, are carried on one deck.

Conductors.—The guides or conductors of a cage exhibit a great variety of practice. In some cases round iron rods screwed together, so as to make one long rod, are stretched from top to bottom of the pit, and are clasped by collars on the cage. This is an old-fashioned practice, and very seldom adopted nowadays; the other kinds of guides being wooden guides, iron or steel wire guides, and iron or steel rail guides.

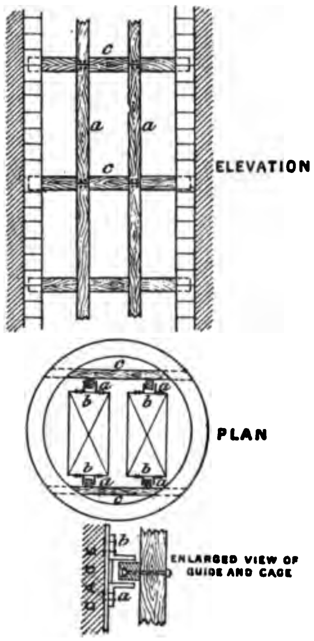


FIG. 536*b*.—Cages and wooden guides. *a*, guides; *b*, shoes; *c*, cross-bearers.

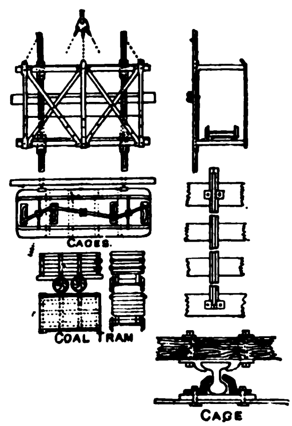


FIG. 537.—Cage, waggons, and rail-guides (Harris's Navigation).

Where wooden guides are used, one is generally placed opposite each end of the cage. A shoe is fitted on the top and bottom of the cage, and on each deck, by which the guide is clasped on three sides (see Fig. 536*b*); the guides are fastened by through bolts to cross-bearers. These guides prevent the waggons from running in and out of the cage, therefore at the top and bottom they are cut away, leaving a sharpened end to enter the shoes on the cage. The cage is guided at the top and bottom by four wooden guides, one at each corner. The guides vary from about 4 inches square to about 6 inches × 4 inches. They must be carefully examined every day from top to bottom. Instead of wooden guides steel guides are sometimes used, shaped like flat-bottomed, single-headed rails (see Fig. 537). Two such rails are fixed vertically down one side of the shaft, and fastened to cross-bearers. On one side of the cage two shoes on each deck clip the rail-head, and the cage is thus guided up and down the shaft. The other cage is similarly guided by rails on the other side of the shaft.

At the Harris's Navigation the guide rails are of steel, weighing 60 lbs. to the yard, each length 27 feet long, bolted to wooden byats fixed in the shaft wall 9 feet apart from centre to centre. These byats are of pitch pine, 9 inches × 6 inches, 13 feet long, on one side of the pit; and on the other side, 17 feet long, 9 inches × 8 inches. At the junctions of the rails the byats are 12 inches × 6 inches, and 12 inches × 8 inches. Each rail-end is bored out to receive a ½-inch dowel. There is also a wrought-iron joint-chair or shoe, clasping the bottom rail, and bolted to the byats. Unless there is some breakage, it is impossible for the cage to get out of its proper track.

Another arrangement is to have wire ropes, or rather rods of thick twisted steel wire, say 7 wires, each ½ inch in diameter, twisted into a strand 1½ inch in diameter, stretched from the top of the pit-frame to the bottom. They are tightened by weights (see Fig. 538) on the bottom of the rod, say 30 cwt. or up to 5 tons on each guide; 4 guides for each cage. At the pit-top the guides go through cross-bearers, above which five or six clamps are secured to each guide to prevent them drawing down. A cast-iron collar (see Fig. 536) bolted to the cage encircles each guide; these guides are, of course, capable of swinging, but with a clearance of 8 inches or more between the cages are found satisfactory. In deep pits, in addition to the guides collared to the cage, a couple of guide rods are generally stretched from top to bottom of the shaft between the cages, so as to prevent the cages from actually coming in contact. Sometimes only two or three guides are collared to the cage, and the other guides in the centre of the pit stand to keep the cages apart. These wire guides are used at depths up to 900 yards; they are very economical, and, requiring no cross-bearers, do not impede the ventilation, and cost but little for inspection and repairs. The steel-rail guides are much in favour, but there is a tendency to cut the head of the rail off by wear. For steady winding it is essential not only to have good guides and collars, but to have a cage well balanced, and to have the centre of the pulley-wheel exactly over the centre of the cage when fixed between the guides, and the centre of the cage must correspond with its centre of gravity.

Lowering - platform. — In winding-cages with two decks, the descending cage is sometimes lowered on to a suspending-platform at the bottom of the pit, and when the loaded tubs have been run on to the bottom deck, the suspending-platform, which has been kept in position by balance-weights and a brake-wheel

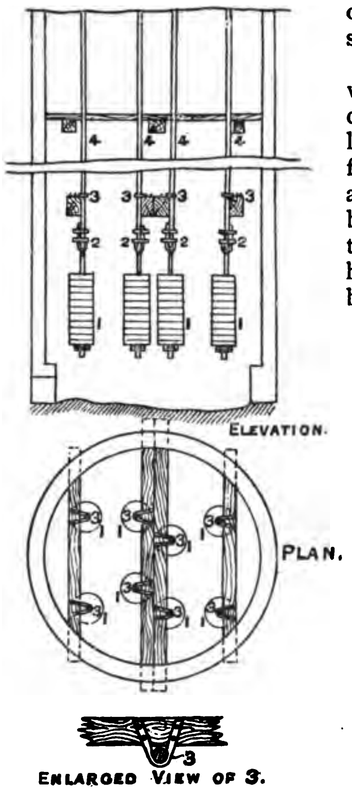


FIG. 538.—Method of fastening conductors. 1, weight; 2, clamps; 3, collars; 4, conductors.

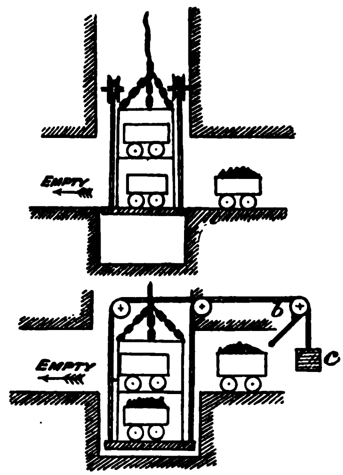


FIG. 539.—Platform and balance-weights at pit-bottom. *b*, brake-wheel and lever; *c*, balance-weight.

(see Fig. 539), is allowed to descend by releasing the brake, and the top deck is then loaded. The balance-weights on the moving platform also serve the purpose of helping to start the loaded cage. The cage on the bank is stopped with the upper deck resting on the props at the same time that the lower deck of the bottom cage is being loaded, and then, when the full waggons have been run off and the empty waggons run on, the lower deck is raised on to the cage-props.

Fowler's Hydraulic Landings.—In some cases movable landings are used, worked by hydraulic power. An apparatus of this description (see Fig. 540) was made and patented by Mr. George Fowler, and is now used at the Denaby Main Colliery in Yorkshire, Cinderhill Colliery, Nottingham, and elsewhere; by means of this, three decks can be loaded at the same time by means of hydraulic pushing-rams, C C. While the cage is running, the movable landings A A A are lowered down by the hydraulic lifting-rams H H and balance-weights G, to the main landing, L L, and then returned to their original position as shown in the figure, ready for the cage again. The movable landings are, of course, at both ends of the cage. There is an apparatus, as shown in the figure, for each cage at the pit-top and pit-bottom. By this contrivance three decks can be loaded as quickly as one.

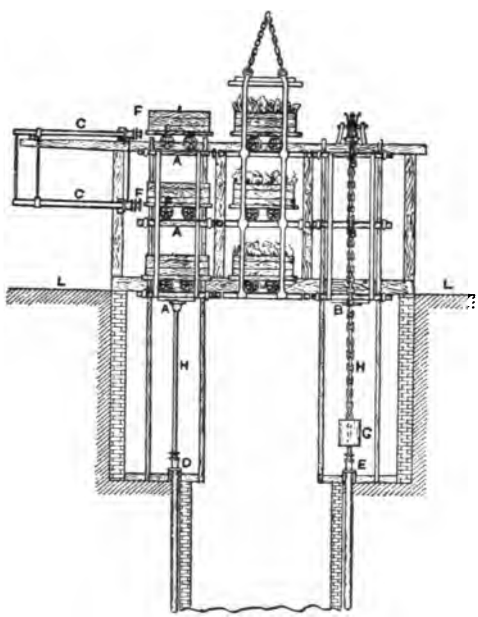


FIG. 540—Hydraulic decks (G. Fowler).

Double Landings.—The pit-bank and pit-bottom are often made with two landings, so that two decks of the cage can be unloaded at once both top and bottom. This arrangement is very suitable for a four-decked cage. If a two-decked cage is used, each deck must be high enough to allow head-room between the landings. Such an arrangement of landings for a four-decked cage is shown in Fig. 541.

Cage-props.—The cage at the pit-top, and at the pit-bottom if there is more than one deck, is supported when standing, on catches, sometimes called props and sometimes fallers; an arrangement of this kind is shown in Fig. 542. There are four catches, *b b*, for each cage, all moved by either of the levers *a a*, one on each side of the pit for each cage, so that there are four lever-handles at the pit. These catches are so arranged that they fall forward by their own weight ready to hold the cage, but if the cage should come up it would readily push them aside if the banksman omitted to withdraw them, though the cage could not descend unless they were withdrawn. As a rule the catches are underneath the bottom of the cage, and are made to catch against the end of it; but sometimes catches are placed on the bank, to catch on the sides of the cage under the top hoop. In some cases catches are not used, the engine-man adjusting the cage to the bank-level without their aid.

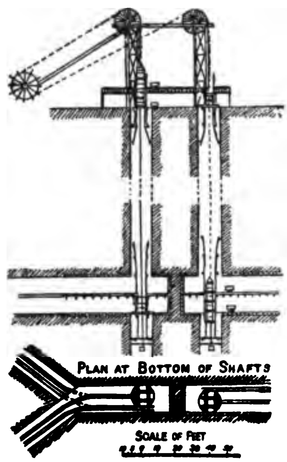


FIG 541.—Double landings.

Unloading Cages.—In order to facilitate the unloading of cages on the pit-bank and at the pit-bottom, one side of the landing is made a little higher than the other, and the rails of the cage are given a little slope, say ½ inch in the length of the cage. This facilitates the moving of the waggons, but this ½-inch slope is not so effective as 3 or 4 inches of slope. But it would not do to have the waggons standing at such an incline on the cage in the shaft, therefore the rails are sometimes made to lift up at one end, and a short iron prop is fastened on to the underside of the rail, which is forced up when the cage is lowered on to the props, so putting the rails on to an incline and causing the waggons to run off.

Another contrivance has been made by Mr. Fisher, of Clifton Colliery, where the empty waggon on the pit-bank is lifted up by a small engine, and so caused to run on to the cage.

Roofing of Bank.—In England the machinery on the bank is, as a rule, only partially covered, though there is a tendency at the present date to protect all the workmen from the rain and snow. On the continent it is customary to cover over the pit-top and pulley-frame with a substantial building, inside which are the pulley-wheels.

Water-winding.—It is often necessary to use winding-engines for raising water. In order that they may do this effectually, there must be a properly constructed barrel or tank of plate iron; such a tank is shown Fig. 543. It is capable of holding, when filled within 6 inches of the top, over 300 gallons. In a large shaft the water-tank may be made much bigger. It is no use having a large tank unless there are large valves by which it can be quickly filled and emptied. In the figure are shown two valves, which open as the tank goes into the water. Upon reaching the top, one of the valves is opened by a lever, B, which strikes against a bar, A, fixed to the pit-frame. Owing to the great size of the valve, the water rushes out with great speed, and the tank is nearly emptied in a few seconds. In a shaft not exceeding 500 yards deep, this tank can be dipped, raised, and

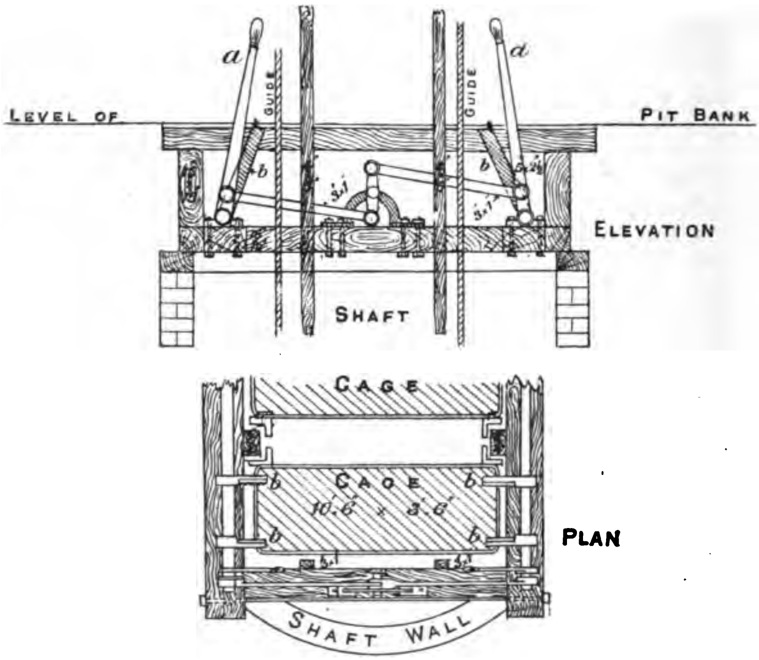


FIG. 542.—Cage-catches.

emptied in about 80 seconds. Thus if there are two such tanks, they will lift together between 400 and 500 gallons a minute. With larger tanks and with a sufficiently powerful winding-engine, 1000 gallons a minute can be wound. A rather ingenious device has been employed by Mr. Galloway, in his sinking at Llanbradach, to fill a water-barrel or tank without dipping it overhead. The tank is round and covered at the top, made of iron plate. At the bottom is a valve by which the water can enter, and by lifting which the tank can be emptied. When the tank has been lowered by the winding-engine to the bottom of the shaft, it is connected by a tube and coupling-joints with an exhausted vacuum vessel in a chamber above the water; by this means the

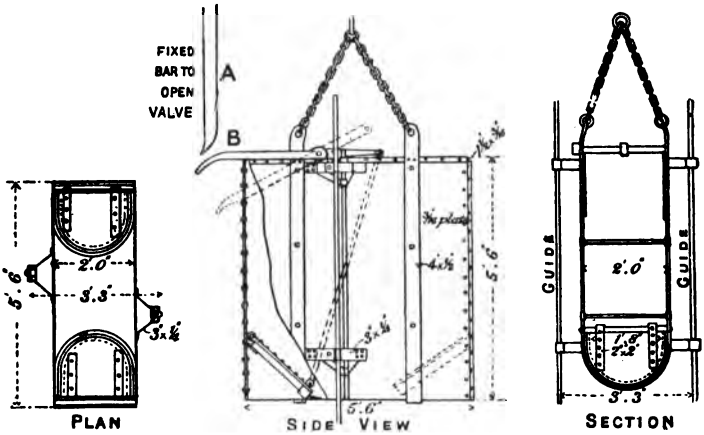


FIG. 543.—Water-winding tank.

water is sucked up into the tank. A small air-pump on the surface keeps the vacuum vessel exhausted of air. The vessel is an old boiler.

CHAPTER XX

"SAFETY-HOOKS" AND "SAFETY-CAGES."

In order to avoid the danger of winding a cage too high, various safety-hooks have been invented by which the cage is disengaged from the rope and suspended at the same time.

Ormerod's Safety-hook.—Figs. 544 and 544*a* show

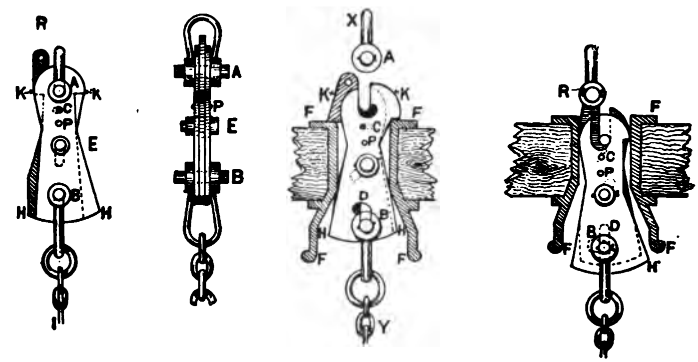


FIG. 544.—Ormerod's safety-hook.

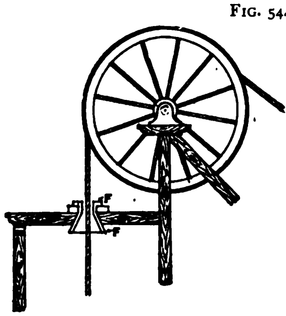


FIG. 544*a*.—Ormerod's safety-hook.

Ormerod's safety-hook. In this case the winding-rope is attached to the link X, and the cage to the chains Y; the iron collar F is fixed in a very strong timber bar just below the pulley-wheel; the winding-rope passes through the centre of this collar. The hook is shown in working order in (1). The upper part will pass through the collar F; but not the lower part, which consists of three plates held together by a centre pin E, and maintained in the position shown in (1) by a copper rivet ½ inch thick, P. When an overwind takes place, the lower shoulders of the plates H H are pressed together, shearing the copper pin P. This causes the shoulders K K to project, so that they cannot fall back through the collar; at the same time the pin of the rope shackle A is brought opposite the slot in the outer plates, and so escapes, whilst the pin B of the lower shackle is brought opposite a slot in the outer plates, into which it falls, but cannot escape. In this way the plates are prevented from resuming their original position. For lowering down again, the rope shackle is attached to the shoulder R on the inner plate, by which it is lifted, the centre plate having a slot by which it can move over the centre pin E after removing the pin C. The hook can then be lowered down through the collar until the cage rests on the bank, when it can be readjusted.

King's Hook.—King's hook is shown in Fig. 545. This consists of four plates. The hook is shown in working order in

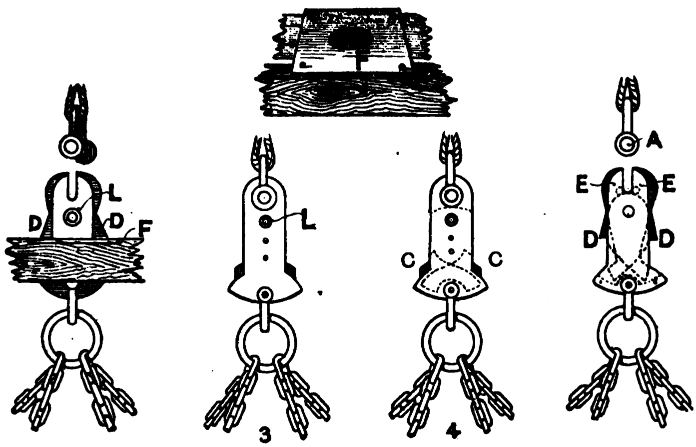


FIG. 545.—King's patent detaching hook.

3 and 4. When an overwind takes place, the shoulders C C are forced in, and the opposite shoulders D D are forced out; at the same time the hooks on the two internal plates E E are forced back from over the pin of the rope shackle A, which is free to escape, but the shoulders D D, catching on the top of the plate F, prevent the hook from returning. The weight of the cage, being suspended from the pin L, keeps the shoulders forced out, and so prevents the cage going down.

Either one or the other of these inventions is in use at nearly all the important collieries in the United Kingdom, and they have never failed except when they have broken. Owing to the

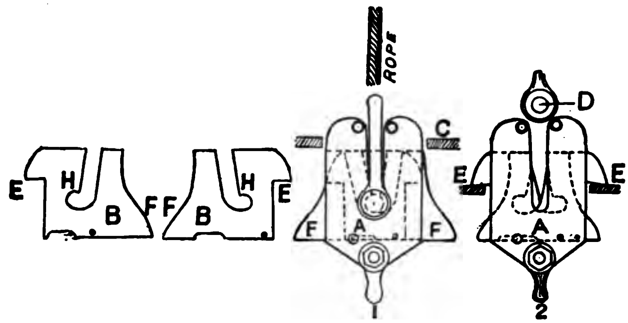


FIG. 546.—West's hook.

great violence with which the overwind takes place, there is some liability to breakage even with the best material.

West's Hook.—Fig. 546 is West's. It differs from the two preceding hooks in this respect: that instead of the plates revolving round a centre pin, they slide. There is an outer casing of steel, A, in which are two slides, B B; the hook is shown in working order in (1). A strong iron plate, C, is fixed in the pit-frame, as in King's, with a hole just large enough for the upper part of the hook to pass through. When there is an overwind, the shoulders of the sliding plates F F are squeezed in, and the upper shoulders E E forced out, so that the hook cannot go back; at the same time the hooks H H, that hold the pin of the rope shackle D, are forced back, allowing the rope shackle to escape, as shown in (2). This is considered a very good hook, and is used at many places.

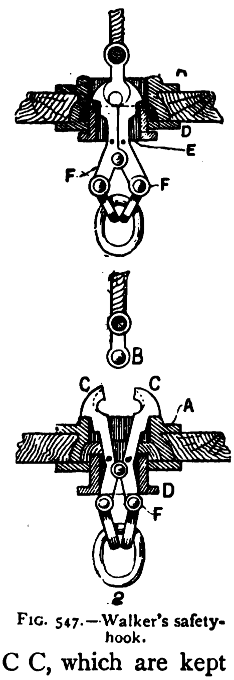


FIG. 547.—Walker's safety-hook.

Walker's Hook.—Another hook, Walker's (see Fig. 547), has an entirely different action. As in the previous cases, there is a collar, A, fixed to strong timbers in the pit-frame; the upper part of the hook only can pass through this collar. The rope shackle-pin B is clipped by the two jaws C C, which are kept tight by the clamp D. When an overwind takes place, this clamp will not go through the collar, but is forced down, shearing the two copper pins E E, and forcing in the two lower arms of the hook F F, causing the upper jaws C C to open, so that the shoulders at the back catch on the top of the collar, as shown in (2). This hook is the lightest in construction of any that have been made, and is in use. It has not, however, met with the same general approval as King's and Ormerod's, several accidents which have occurred having been attributed to some defect in the principle of its construction. It is, however, adopted by many engineers of great experience in preference to any other hook.

Safety-cages.—These are intended to obviate the danger arising from a broken rope. They are only partially adopted in Great Britain, though they are generally adopted on the continent, where, however, safety-hooks are not in such general use as in England.

Figure 548 shows a safety-cage (Owen's). The construction will be readily understood. The toothed

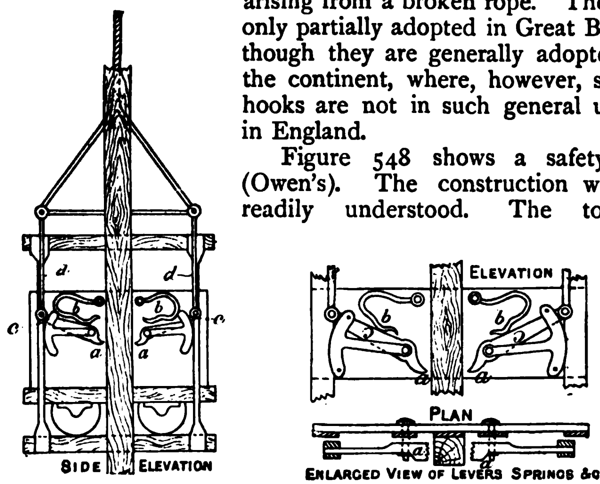


FIG. 548.—Owen's patent safety-cage.

clutches *a a* are pressed towards the conductors by the springs *b b*; the levers *c c*, attached to the toothed clutches, are connected to the winding-rope by the rods *d d*, so that when the cage is suspended, the levers compress the springs, and force the clutches back from the conductor. In the case of the rope breaking, the cage is, of course, no longer suspended, and the clutches are forced against the conductors by the springs; the falling cage tends to force the clutches still more tightly against the conductors, and so it is stopped. There are many varieties of safety-cage constructed on the same principle; one of these is shown in Fig. 549. In this case the clutches *a a* are held in position by the inclined iron channels *b b*, in which they slide. They are forced against the wooden guide-rod *c c* by means of the volute springs *d d*. When the cage is suspended, the rope and chains pull up the rods *e e*. On each of these is a shoulder which compresses the springs against the top of the cage. Upon the rope breaking, there is no longer any strain to compress the springs, which thereupon expand, forcing down one end of the lever *f*, and forcing up the clutches *a a* against the rods. The figure shows one end of the cage; there are, of course, similar clutches at the other end.

Another safety-cage, Calow's, is shown in Fig. 550. The principle of this is highly ingenious. There is a toothed clutch, *a*, keyed on a shaft, S, which is forced against the conductor *b* by the spring *c*, pressing upward under the lever D, also keyed on to the shaft S; but the action of the spring is resisted

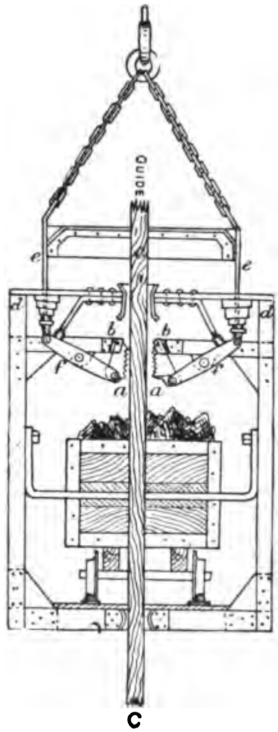


Fig. 549.—Hasard safety-cage.

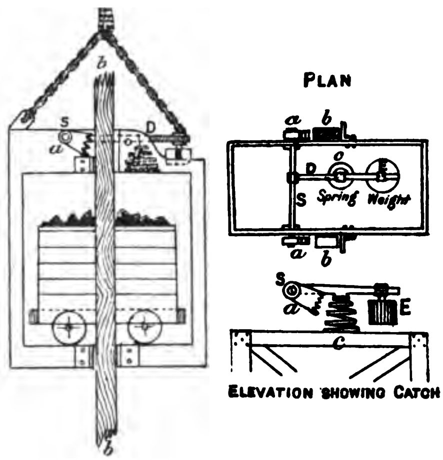


Fig. 550.—Calow's safety-cage.

by the weight E, so that until that weight is removed the clutch cannot grasp the conductor. If the cage, however, were free to fall, then the weight E would not press upon the lever D, because all the parts would be falling together. The instant the rope breaks the cage is free to fall, therefore the weight E ceases to be a weight in regard to the lever D on which it rests; consequently the spring *c* acts and forces the clutch against the conductor. The falling cage tends to jam the clutch still tighter, so that when the cage stops the weight E has no power to reopen the clutch. This apparatus has been used at a good many collieries.

CHAPTER XXI.

PUMPING-ENGINES: CORNISH, COMPOUND, HORIZONTAL, FLY-WHEEL, HYDRAULIC, PNEUMATIC, ELECTRIC.

No class of machinery is more important to the miner than pumping-engines, which are necessary for draining the mine; in fact, the first efforts of the inventors of steam-engines in this country seem to have been directed to the construction of a pumping-engine. Savery's engine, called the miner's friend, acted in a manner very similar to that of the modern pulsometer. Steam from the boiler was admitted by means of a hand-tap into the condenser; as soon as the condenser is full of steam, the tap is shut; the steam condenses and the water rises up through a suction pipe to fill the vacuum; then the steam-pipe is opened, and the pressure of the steam acting on the surface of the water forces it out through the rising pipe or main. The steam-tap is again shut, and a jet of cold water out of the rising main is allowed to fall on the condenser; as the steam is condensed the water rises up the suction-pipe again; the steam-tap is opened, and the water forced out into the rising main. This pump would only suck the water a short distance, perhaps 20 feet, and force it to a height corresponding to the steam-pressure, say another 20 feet, or perhaps more if the pressure was high. The idea was to place one engine above another in the shaft-side, so forcing the water to the top in a manner exactly similar to that done with the pulsometer shown in Figs. 159 and 160 already described.

Newcomen Engine.—The Newcomen engine was like the modern steam-engine, having a cylinder, piston, beam, etc. This was improved by Smeaton, and engines similar to those made by Smeaton have been at work within the last twelve years. It is a vertical cylinder beam-engine; beneath the cylinder is a pot-condenser. The cylinder is open at the top, and is single acting, that is to say, the pressure only takes the piston one way, that is, downwards, due to the weight of the atmosphere. The piston is connected to the beam, and the outer end of the beam to the pump-rods, by means of chains, a cross-piece, cut to the segment of a circle, being fixed on to each end of the beam, so that the movement of the piston and pump-rod is vertical.

Boulton and Watt.—The Boulton and Watt engine is an

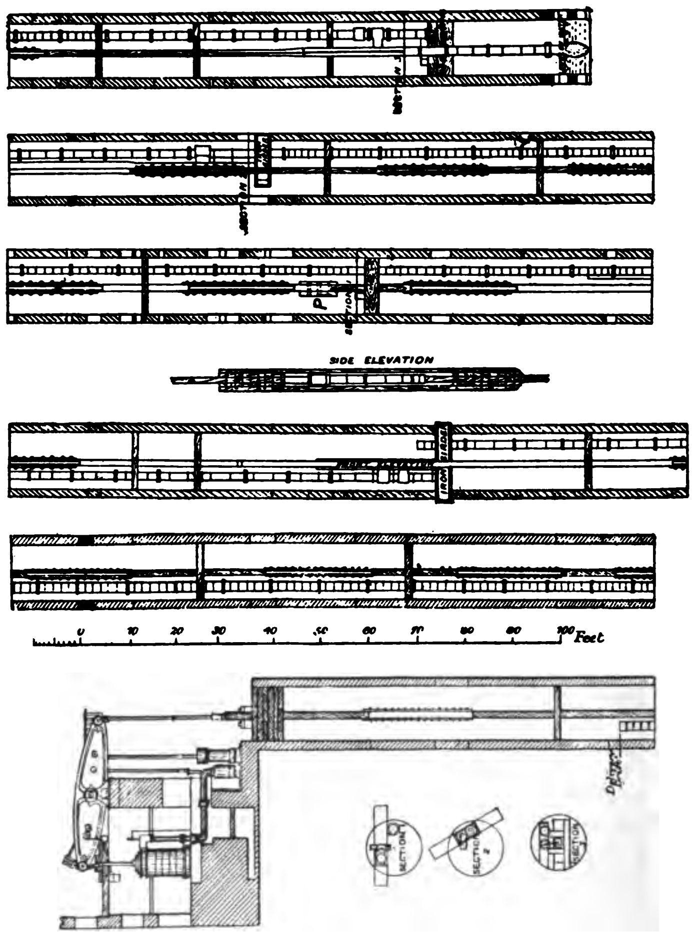


FIG. 551.—Cornish engine and pumps in shaft.

improvement upon the Newcomen. The modern type has a covered cylinder and four valves, so that steam is admitted at each end of the cylinder in turn, the exhaust-pipe leads to a separate condenser, from which a passage leads to the air-pump.

This engine is sometimes used to give movement to a crank and fly-wheel, and thence by connecting-rods to beams over the pit; and sometimes the outer end of the engine-beam is directly over the pit, and connected to the pump-rods.

Cornish Pumping-engine.—The Cornish engine (see

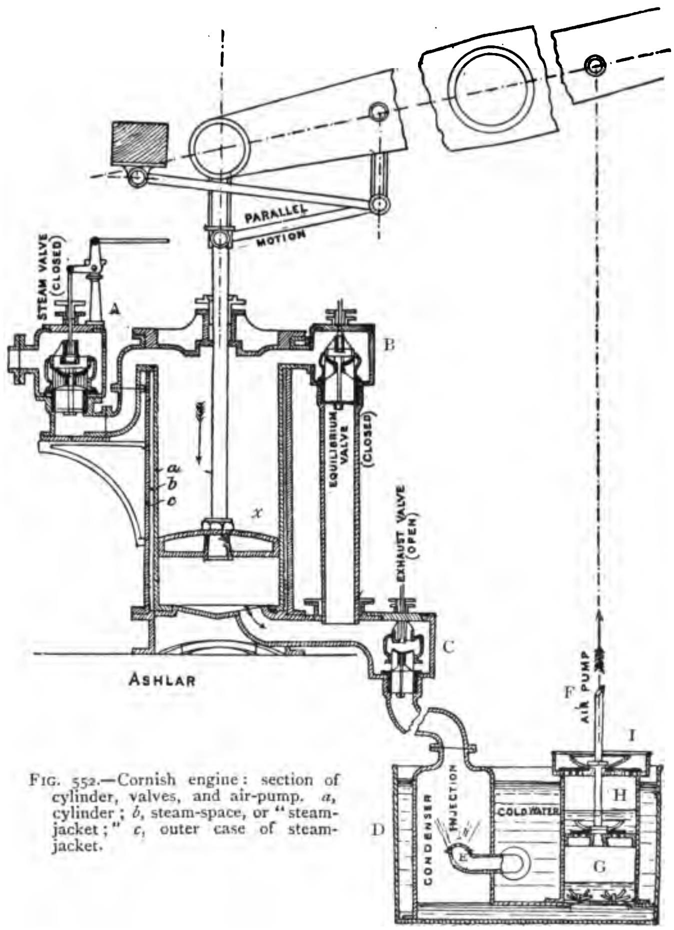


FIG. 552.—Cornish engine: section of cylinder, valves, and air-pump. *a*, cylinder; *b*, steam-space, or "steam-jacket;" *c*, outer case of steam-jacket.

Figs. 551–553) is made on the principle of Watt's engine, but differs from the modern Boulton and Watt in having only three valves and being single acting. Steam is admitted on the top of the piston for the downstroke; when this is completed the equilibrium-valve at the top of the cylinder is opened, and the steam from the top passes through a pipe to the bottom of the cylinder, thereby placing the piston in equilibrium. The upstroke is made by the weight of the pump-rods attached to the outer end of the beam, and the steam forced from the top of the piston to the under side. The exhaust-valve is then opened, letting the steam from under the piston into the condenser; at the same time, the steam-valve is opened, admitting a fresh supply of steam to the top of the piston, the downstroke being made partly by the steam-pressure, and partly by the vacuum in the condenser. The piston-rod is guided by parallel motion. A Cornish pumping-engine is generally considered one of the best kinds; its economy, however, depends upon the observance of well-known rules for the economical working of steam. The boiler, steam-pipes, and cylinder are carefully covered over with non-conducting material to prevent the radiation of heat; in some cases the cylinder has a steam-jacket. The steam is used expansively; that is to say, the steam-valve is shut at say from $\frac{1}{3}$ to $\frac{2}{3}$ of the stroke, the rest of the stroke being performed by the expansion of the steam, the suction of the vacuum, and the momentum of the moving parts. The engine stops at the end of every upstroke of the piston, and does not start again until by the operation of the cataract a weight is set free, which in falling opens the steam-valve; the piston then goes down, and is drawn up again by the weight of the pumps. The cataract is shown in Fig. 554; it consists of a little ram-pump. The ram G is lifted up by the beam *e* by the action of the plug-rod attached to the engine-beam on the lever *a* as the piston goes up, and it sucks water from the tank *h*. The ram G is free to fall again, except for the water underneath it, which can only escape through a small tap, K. The ram falls by the action of the weight *l*, at a speed corresponding to the opening in this tap, which can be regulated by the rod *o*, so that the ram may fall in a second or in a minute. When the ram has nearly reached the bottom, the pin *m* on the beam *e* lifts the rod *n*, and so releases a sector on the horizontal

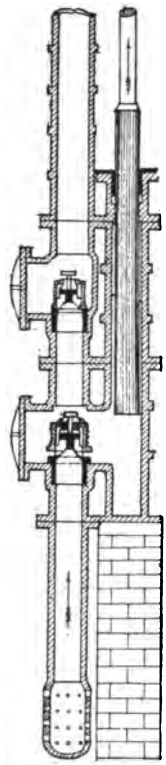


FIG. 553. — Cornish engine: section of pump and valves.

axis, thus setting free a heavy weight which has been lifted up by the engine; the falling of this weight by means of a lever opens the steam-valve. A Cornish engine is generally made with a stroke of from 8 to 12 feet, and with a cylinder from 40 inches to 110 inches in diameter. A large engine is one with a cylinder

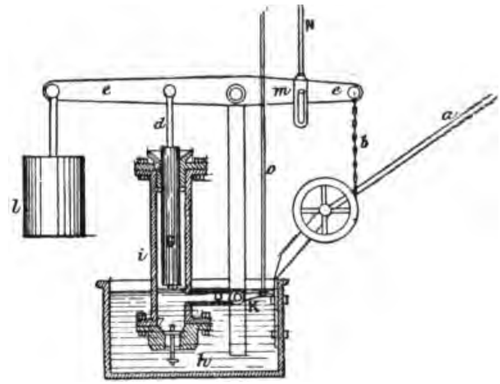


FIG. 554.—Cataract.

84 inches in diameter, and 11-feet stroke. The pressure in the steam-boiler is from 20 to 60 lbs., and the number of strokes per minute from four up to eight. Six strokes is a good speed for an engine of which the piston travels 10 feet each way; when six strokes a minute is spoken of, it means that the piston goes up and down six times, and if it goes 10 feet each way, it moves 120 feet in a minute. Fig. 552 shows a section of the cylinders, valve-chests, condenser, and air-pump. A is the steam-valve admitting steam on top of piston x; B is the equilibrium-valve, allowing steam free passages from top of piston to the under side; C is the exhaust-valve connecting bottom of cylinder with the condenser D; E is the injection of water which condenses the steam, and F is the air-pump cylinder; G is the suction-valve at bottom; H, the bucket or piston with valves; and I is the cover with delivery-valves. As a rule, the water on the bucket strikes this cover on

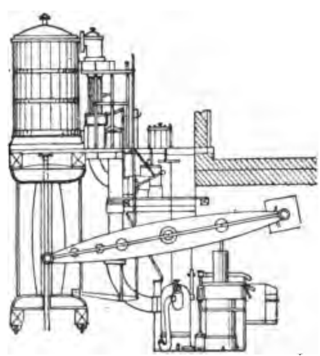


FIG. 555.—Bull engine.

the upstroke, and it is therefore made to lift, so as to reduce the shock.

Bull Engine.—The Bull engine (see Fig. 555) differs from the Cornish engine in having the cylinder placed directly over the shaft, the piston-rod of the engine being connected directly to the

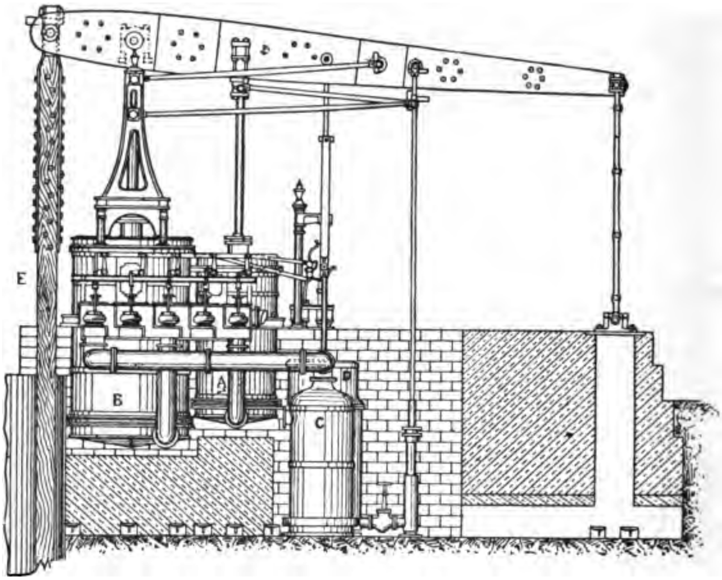


FIG. 556.—Barclay engine. A, high-pressure cylinder, 51 inches diameter; B, low-pressure cylinder, 73 inches diameter; C, condenser; D, air-pump; E, pump-rod. Stroke of A, 7 feet 9¼ inches; stroke of B, 10 feet.

pump-rod. By means of a long connecting-rod, a balance-beam is attached to the cross-head of the piston-rod, and a counter-balance weight is at the other end. Rods from this beam work the valves, air-pump, etc. This engine is preferred by some engineers, but the pit-top is less accessible with this engine over it than with the beam-engine.

Barclay's.—Another arrangement is that adopted by Barclay. The cylinder is between the pump-rods and the fulcrum of the beam. By this arrangement the stroke of the pump-rods is rather longer than that of the engine (Fig. 556).

Horizontal Engine.—As in winding-engines, so in pumping-engines, the engine is often laid horizontally. A connecting-rod from the cross-head gives movement to the pumps at the pit-top (see Fig. 557[1]). The horizontal engine may be similar to a Cornish

[1] J. B. Simpson, N.E. Inst.

engine, or it may have a fly-wheel; in this latter case the piston-rod comes through the back cylinder-cover, and by means of a connecting-rod gives movement to a fly-wheel. The fly-wheel has the advantage of regulating the length of the stroke, so that the

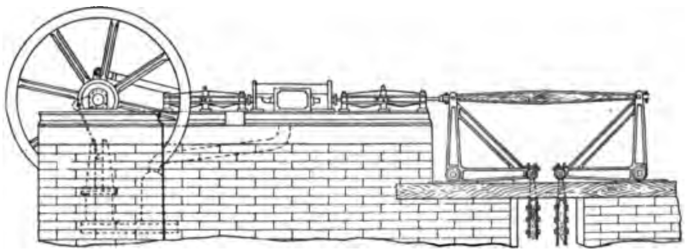


FIG. 557.—Horizontal engine with fly-wheel.

piston must always make a stroke of exactly the same length, and is therefore less likely to strike against the cylinder-cover. Some engineers prefer pumping-engines with fly-wheels, and some prefer them without; but it is, however, a fact that some of the finest examples of pumping-machinery in the country have fly-wheels, and that other equally fine pumping-engines have no fly-wheels.

Compound Pumping-engines.—The economy with which steam is used depends, amongst other things, on the degree of expansion. Steam may be expanded twenty times in a steam-engine with advantage, if the original pressure is high, say 100 lbs. or more, and there is a condenser. This high degree of expansion cannot be safely obtained with a single cylinder, because the admission of steam at a very high pressure to a large cylinder puts a dangerous strain upon the machinery. This difficulty is got over by the adoption of two or more cylinders. The steam is first admitted into the smaller cylinder, where it is cut off at one quarter or half stroke; the exhaust steam from this cylinder is then taken into a large cylinder, say twice the diameter and four times the capacity; the steam is here expanded four times again, and as it was expanded twice or four times in the high-pressure cylinder, the total expansion is eight or sixteen times. This method of expanding is used with vertical engines by placing the two cylinders side by side, as in Fig. 556, and in horizontal pumping-engines by placing one cylinder behind the other, as in Fig. 564.

Arrangement of Pumps in Shaft.—Fig. 551 shows the method of arranging the pumps attached to a Cornish pumping-engine. It will be seen that there are three rams one above the other, each ram being carried on beams fixed across the pit. Below the bottom ram is a bucket-pump; this bucket-pump fills the bottom ram cistern, the bottom ram fills the middle one, and the middle one the top ram cistern. Water running down the shaft-sides is conveyed into the cisterns from which the rams take their supply. It is essential that there should always be sufficient water in the cistern to supply the ram on its upward stroke; if this is not so, and the ram-cylinder should only be filled say three-fourths of the distance, the pump-ram on the downstroke would fall down on to the top of the water rapidly, and when it struck the water there would be such a concussion that the machinery would very likely be broken and the ram-case burst. In order to keep the latter "solid," to use the technical phrase, the lower pump is always bigger than the pump above, unless there is a feeder of water in the shaft-side going to the upper pump; any surplus of water raised by the lower pumps is carried down the pit by an overflow-pipe. The reason for placing a bucket-pump at the bottom of the pit is that, in case of flooding, the ram and valves connected with it could not be got at for repairs if it were underneath the water; but the bucket and clack can be drawn by means of the rods and repaired when they are under water. If the rods should break at the top they would not fall down the pit, because clamped to each side of the rod at intervals are packing pieces of wood, and beneath these are strong timber beams fixed into the shaft side, which prevent the rods from falling further than their proper stroke; the rods are also guided by cross-bearers in the shaft (see P, Fig. 551) against which the rods may rub, and by which they are prevented from bending. In order to prevent the rods being worn out by friction against the guides, boards are fixed on to the side of the rod, springing a little way out from the rod at the centre, so that, instead of the rods touching these guides, it is these outside boards or rubbers, and they yield when pressed against the guides, so as to prevent any shock. The proper working of a Cornish pump depends upon having the weight of the rods properly balanced. The rods and pump-rams are always heavier than the column of water, and it is this excess of weight which causes them to go down; but the weight is generally too great for this purpose, and a balance-beam is placed at the pit-top with a balance-weight put in a box. The amount of this balance-weight can be altered as required, the faster the engine has to work the less the balance-weight. But where the pit is very deep one balance-beam is not sufficient, and one or more balance-beams are placed in chambers in the shaft-side and attached to the rods by long connecting-rods (see Fig. 558), so that each length of say 100 yards of rods has a separate balance-beam; this reduces the strain upon the rods at the pit-top. Great attention has to be paid to the proper balancing of the pumps if they are to work at a good speed without undue shock. Where the shaft is inclined, as in many metalliferous mines, the arrangement of pumps is similar to that just described, only the pump-rods rest upon rollers fixed on the shaft-side.

Pump-valves.—Many kinds of valves are used. One is the ordinary butterfly-clack as described in sinking-pumps. Sometimes, instead of the valves being hinged at the centre, they are

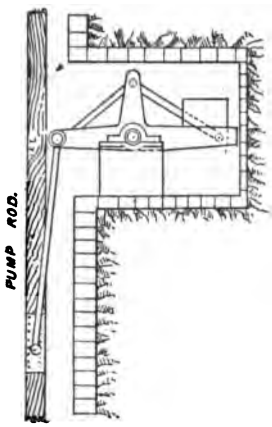


FIG. 558.—Balance-beams on shaft-side.

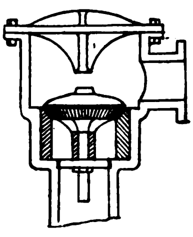


FIG. 559.—Mushroom-valve.

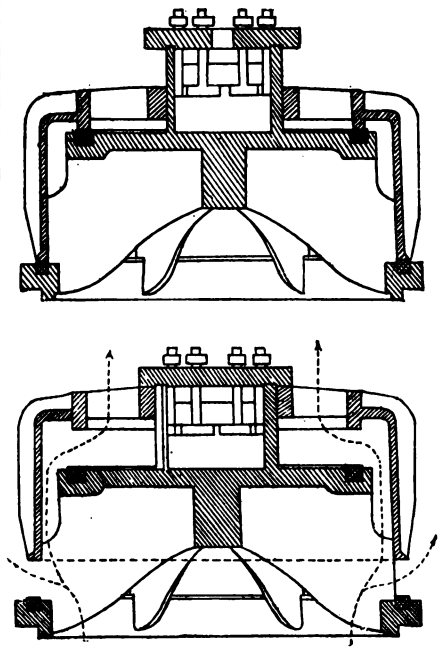


FIG. 560.—Double-beat valve with gutta-percha seats.

hinged at the side of the clack-shell, opening at the middle; and they are often made so that the whole valve can lift, the spindle of the hinge moving up and down in a vertical slot. Instead of butterfly-valves a mushroom-valve is sometimes used (see Fig. 559); this is simply a round plate of iron or brass shaped like a mushroom, the central spindle working in a collar. In order to reduce the shock caused by the closing of a large valve, double-beat valves are generally used (see Fig. 560). These valves are ring-shaped, and have two seats or beats one above the other; the two seats are in one casting, and the centre of the inner seat is filled with a solid plate, so that no water can pass except between the valve and the seats. When the valve is lifted, water passes under it at the inner and outer ring; there are thus two passages for the water under the valve as compared with only one passage where a simple mushroom-valve is used. For this reason the valve need not be lifted so high in order to give water-space, and as the valve is not lifted so high, it does not come down so fast when it closes, and therefore there is less shock. There is also less weight upon it, because it has only to bear the weight of a small ring of water. The average diameter of this ring is the average of the diameters of the two seats, and the width of the ring is the difference between the inside diameter of the upper beat and the outside diameter of the lower beat, and this need not be more than 1 inch; therefore the only weight upon this valve is that due to this ring 1 inch wide, so that both the valve and the valve-seat last longer than they would do if they had to bear the whole weight of water. But where metal beats upon metal there is a tendency for the valve to get injured and worn out. This is sometimes remedied by introducing a narrow ring of leather or of gutta-percha into the valve-seat, as shown in Figs. 560 and 561. If this is well made, it will stand a great deal of wear.

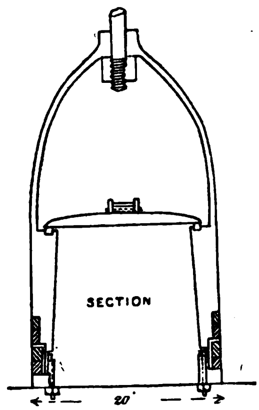
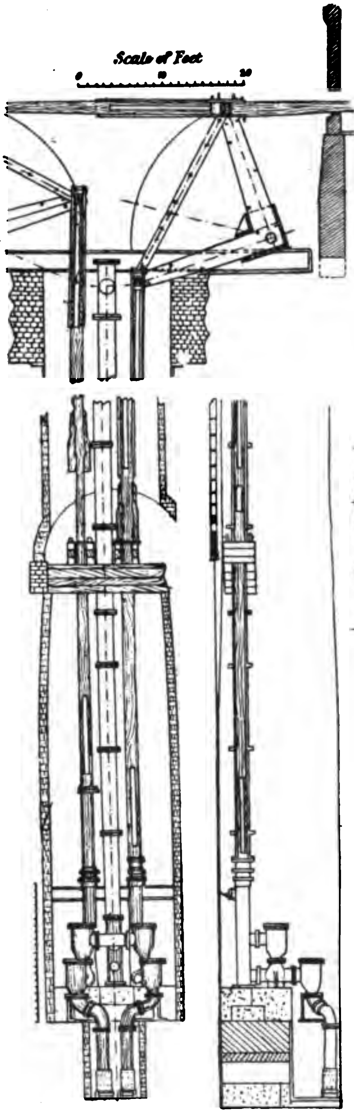
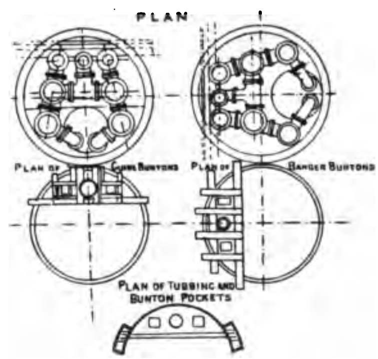


FIG. 561.—Steel bucket with brass rings.

Long Lifts.—Of late years it has become common to have long lifts. Sometimes a bucket-lift is made to lift 100 yards and more, but where there is a great depth there is great wear and tear of the buckets. Fig. 561 is a drawing of a steel bucket used for a lift of 100 yards, 20 inches in diameter. The bucket is packed with brass rings. The valve at the top is steel lifting all round, but held in its place by a hinged spindle moving in vertical slots at one side; it beats upon a ring of gutta-percha. Where a ram pump is used, there is no limit to the height which the water may be forced. At the Staveley Collieries (see Fig. 562) there is a compound horizontal steam-engine without fly-wheel (by Hathorn Davey and Co. of Leeds), described by the late Mr. Humble at the Chesterfield and Midland Counties Institute. This engine gives movement to two beams with L legs; to each of these beams is attached a pump-rod of pine 16 inches square. The rams are at the bottom of a pit 240 yards deep, and are each 20 inches in diameter; they each force water up one central

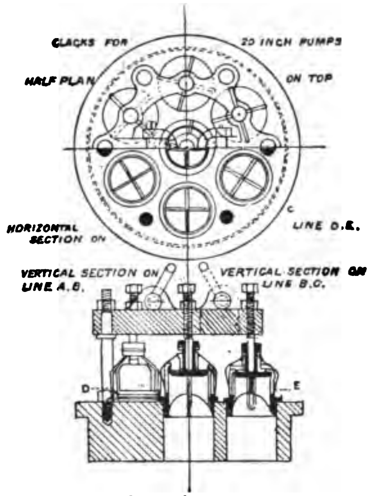




	ft.	in.
Diameter of rams	1	8
Stroke of rams	9	8
Diameter of rising main	1	6
Height of column	740	0
Diameter of high-pressure cylinder	3	9
Diameter of low-pressure cylinder	6	0

FIG. 562.—Staveley ram pumps.

rising main 18 inches in diameter. As the pressure upon the valves at this depth would be very great, instead of having a large valve, seven double-beat valves 5 inches in diameter were used (see Fig. 563). By this means the pressure per square inch of valve-seat was reduced to a comparatively small figure, and at the same time a larger water-way was provided with a less height of valve-lift.



Area of each valve under pressure of column	16·9 sq. in.
Total area of openings in the seven valves for delivery . . .	222·6 ,, ,,
Weight of clack	7 lbs.
Pressure per square inch to lift clack	0·777 lb.
Diameter of pumps . . .	20 in.

FIG. 563.—Staveley ram: small valves.

Underground Pumps.—It has become quite common to place a pumping-engine at the pit-bottom, or only 5 or 6 yards above the pit-bottom. This engine sucks the water out of the sump and forces it to the top in one column. This plan is adopted because the steam-pipes take up less space in the shaft than pump-rods, and also because in many cases there is no space at the pit-top for the erection of a pumping-engine. From some points of view, the pit-bottom is not the right place for a pumping-engine, because, if it should break down, or be overpowered by an additional feeder of water, it might be drowned, and nobody could then get at it to start it. Underground pumps, however, are generally used as supplementary to a pump at the surface, or where the winding-engine is capable of raising the water should the pump break down, or else the pumps are provided in duplicate. The chief difference between an underground pump and a surface pump is, that the former has its ram attached to the piston-rod of the steam-engine direct, instead of through a long length of rods; and, for convenience of erection underground, these engines are generally horizontal. They are made both with and without fly-wheels, and there is a great variety. Fig. 564 shows one with fly-wheel, which is similar to one recently erected by the writer (except that in the figure the engine is shown compound) which works satisfactorily, and Fig. 565 shows a pair of pumps like a "Worthington pump." Underground pumping-engines are often worked by compressed air, which acts in the same way as steam. Compressed air, however, on its expansion produces great cold, and in some cases the exhaust ports are filled up with ice, bringing the engine nearly to a stop. This difficulty has been remedied in several ways. One way is to round off the edges of the ports. It has been found in some cases that the ice does not then adhere. Another plan is to have a gas-burner or an oil-lamp to keep the exhaust port warm. Hot irons are sometimes sent down the shaft from the boiler fire, and placed upon the valve-chest. A better method is to pass the compressed air through a small heating-stove, which enhances its economy of use. It is said by some engineers, that if air is expanded so as to be "exhausted" at a pressure below that of the atmosphere, no ice will be formed. Underground pumping-engines are also driven by hydraulic pressure forced through pipes from a pumping-engine at the surface. In some cases an underground steam-pump is

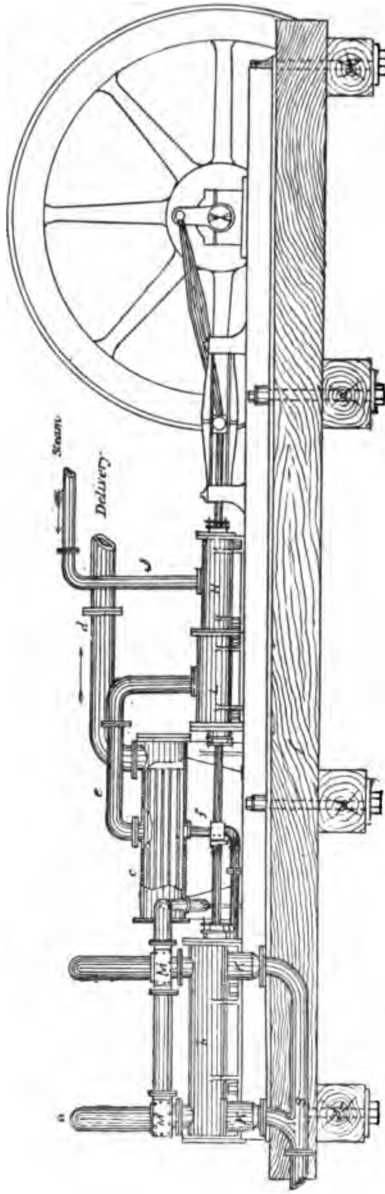


FIG. 564.—Compound steam-pump, underground, with fly-wheel. *a*, air-vessels; *b*, pump; *c*, condenser; *d*, delivery-pipe; *e*, exhaust to condenser; *f*, air-pump suction-pipe; H, high-pressure cylinder; L, low-pressure cylinder; J, steam-pipe; K, suction-valve boxes; M, delivery-valve boxes; S, suction-pipe.

fixed in the shaft, say 50 or 100 yards above the bottom (see Fig. 566), and it is supplied with water by a hydraulic pump at

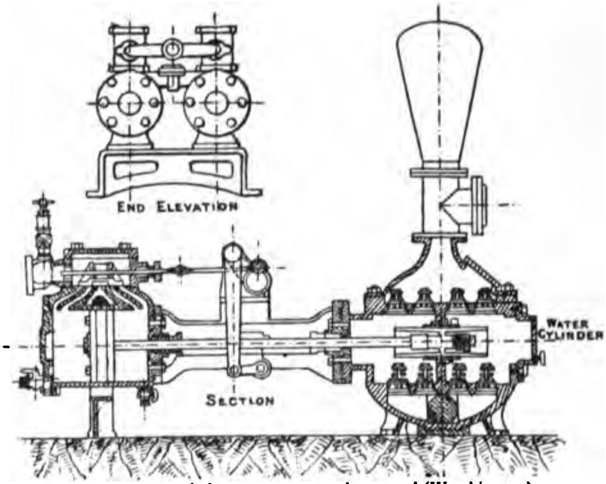


FIG. 565.—Coupled steam-pump, underground (Worthington).

the pit-bottom, which itself is driven by the pressure of water from the rising main of the steam-pump above. Electric pumps have been introduced of late years. The arrangement of one, as erected by Mr. Frank Brain at the Trafalgar Colliery, is shown in Fig. 567. This is one of the first, if not the first, electric pumping-plants in the mines of this country.

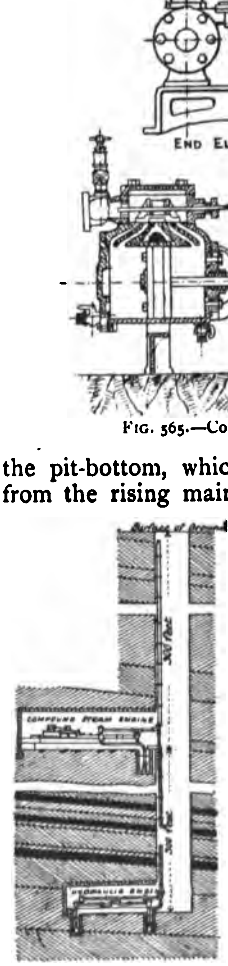


FIG. 566.—Steam and hydraulic pumps in pit.

A fine example of underground hydraulic pumping-machinery is shown in Figs. 568 and 569. This was erected at the silver-lead mines of Clausthal, and were new when seen by the writer in 1883. The engines are at a depth of 680 yards below the surface; the water-supply is obtained from reservoirs on the hills above the mine. There is an adit driven from a valley 7 miles distant, which reaches the shaft at a height of 260 yards above the pumping-engine; the pumping-engines are thus driven by water having a head of 680 yards, and have to raise the water to a height of 260 yards. The difference in pressure, 420 yards,

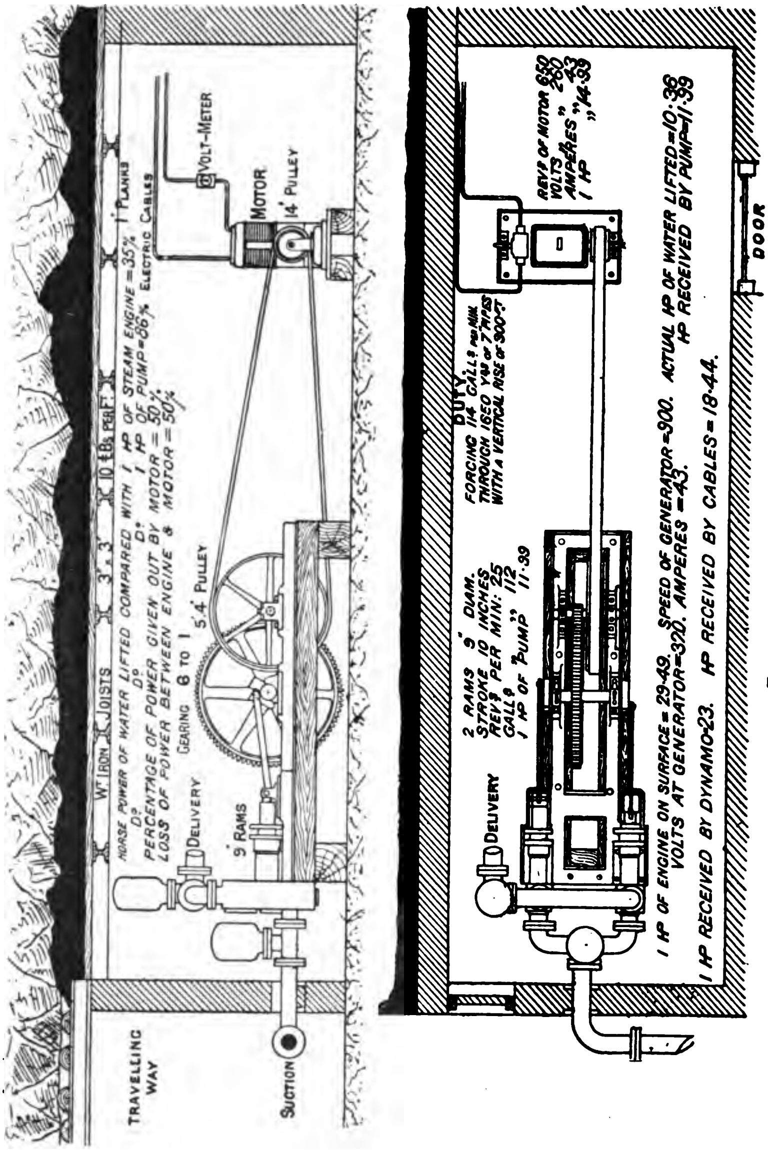


FIG. 567.—Electrical pumping in mines.

is equal to about 540 lbs. per square inch. Each engine has a pair of horizontal cylinders 12 inches in diameter, with a stroke of 2 feet; the piston-rods, 6½ inches in diameter; the pump-

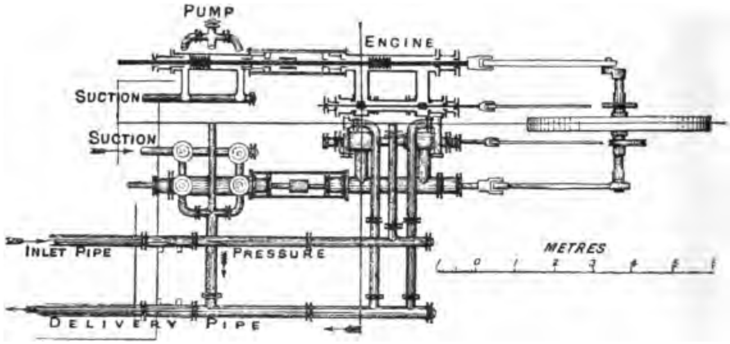


FIG. 568.—Clausthal hydraulic pumps.

cylinders, 13 inches in diameter. The cylinders, rods, and pumps are all brass. This pair of engines is duplicated by another on the other side of the shaft. The total cost of the installation was £16,000. These particulars were obtained by the writer on the spot; the figure is reduced from a drawing prepared by Professor Oscar Hoppe.

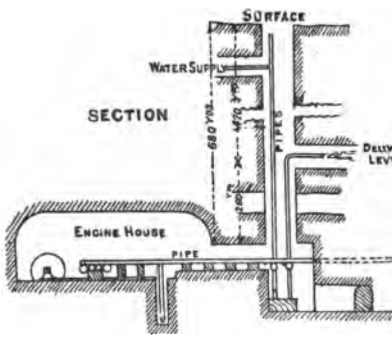


FIG. 569.—Clausthal hydraulic pumps: shaft, and hydraulic column.

Another fine example of hydraulic pumping-machinery was erected in Derbyshire at the Allport Lead-mines. The water was brought from a river at some distance, and had a pressure of 58 lbs. on the square inch at the top of the mine, where the engine was erected. This was direct-acting; the cylinder 50 inches in diameter, 10-feet stroke. The water was raised a total height of 132 feet by a plunger 42 inches in diameter; the pump-rod was 20 inches square; the pump-trees were 40 inches in diameter; and, in case the mine should be flooded and the ram inaccessible, provision was made for inserting a bucket 38 inches in diameter. The engine would work without undue concussion seven strokes a minute, and at that rate would raise more than 4000 gallons a minute. Its usual speed was five strokes.

Pumps placed at a great distance from the shaft in dip workings are often driven by endless ropes, the rope driving a pulley, and that, by means of gearing, giving movement to a revolving shaft, cranks on which drive the pumps (see Fig. 570).

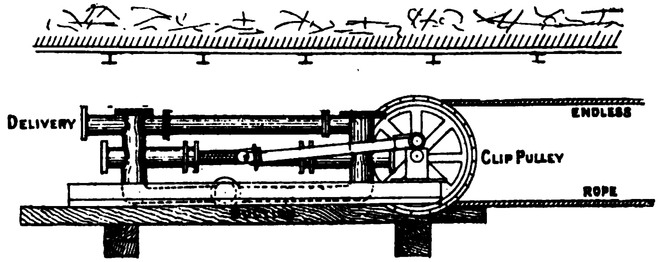


FIG. 570.—Rope-driven pump.

Useful Effect of Pumps.—In a well-arranged vertical pumping-engine, working in a vertical shaft about 100 yards deep, 80 per cent. of the power of the engine may sometimes be found in the water lifted. The power of the engine is ascertained by an indicator in the method already described for fan engines. The work done in raising the water is ascertained by taking the amount in gallons, and multiplying that by 10 to turn it into pounds (a gallon of water weighs 10 lbs.), and multiplying the weight by the height to which the water is raised in feet: thus 1000 gallons a minute raised 100 feet = 10,000 lbs. raised 100 feet = 1,000,000 foot-pounds, and this divided by 33,000 gives about 30 H.P. If, instead of 100 feet deep, the mine had been 400 feet deep, the horse-power in water would have been 120, and if that is 80 per cent. of the engine-power, it follows that the indicated horse power of the steam-engine required would be 150 The efficiency of the pump will, of course, be less with a long length of rods than with a short one. Each beam and bell-crank absorbs some power in friction; horizontal engines and pumps have more friction than vertical ones. The efficiency in some cases is only 50 per cent. In the case of some large underground pumps raising 1500 gallons a minute over 800 feet high, the useful effect is said to be 88 per cent. The larger the pump, and the higher the lift, the greater the useful effect. The writer observed a horizontal underground pump, with fly-wheel, about 15 H.P., that had a useful effect of 50 per cent. He also observed a vertical, direct-acting pump, without fly-wheel, of about 20 H.P., that had a useful effect of 90 per cent., the difference in the efficiency being entirely due to difference in construction, the second engine having less friction. The fuel consumed by a pumping-engine depends on the quality of the boilers, engines, the pressure, degree of expansion, etc., and varies in actual practice from about 2 lbs. of coal per hour per indicated horse-power of engine up to 20 lbs.; but a moderately good plant consumes about 5 lbs. of good slack per I.H.P.

Strength of Pipes.—The pipes, clack doors, and bucket door-piece have to be made very much stronger than would appear necessary to bear the pressure of water inside, because the water is continually rising and falling, and sometimes it jumps in the pipe and falls back with a shock, calculated to burst the pipes, unless there is a very large margin of strength. The shock at the end of a stroke in a bucket pump sometimes produces a pressure equal to four times that due to the statical pressure. Thus, if the height of the lift is 60 yards, the pressure is say 78 lbs. per square inch on the top of the bucket. This pressure may be increased by the water falling back at the end of the stroke to 312 lbs. per square inch, and this pressure will operate upon the whole interior surface of the bucket door-piece. The factor of safety should be about five, so that the dimensions of the door-piece must be calculated to resist an internal strain of 1560 lbs. on the square inch. The whole of this strain has to be borne by the parts above and below the bucket door, because whatever strength is gained by the bolts on the door must be regarded as additional security. Fig. 571 gives the strengths and dimensions of the pumps of a lifting set.

Air-vessels (see Fig. 571*a*) are often used for underground pumps, the engine forcing the water through a pipe underneath the air-vessel. With every stroke of the pump the water rises in the air-vessel, compressing the air; this takes place at that part of the stroke when the pump-ram is delivering its maximum quantity of water. When it is delivering less, the expansion of air forces some water out of the air-vessel, thus keeping up a uniform flow of water, as well as reducing the strain on the pipes by preventing shocks either from the ram or the moving column of water. An air-vessel, to be efficient, must be kept at least two-thirds full of compressed air at the pressure due to the head of water, and the size of the air-vessel must be not less than six times the room occupied by the water delivered by each stroke of the ram. Sometimes an air-vessel is placed in the rising main of a vertical Cornish ram pump. An air-vessel, properly managed, is of the utmost importance wherever there is a single ram forcing through a long length of pipes. Every air-vessel should have a glass gauge to show the height of the water, and an inspirator, or other kind of air-pump, to force in the requisite supply of air to keep the air-vessel properly charged with compressed air.

Man-engine.—In many metalliferous mines the workmen ascend and descend the shaft by ladders. In some cases they

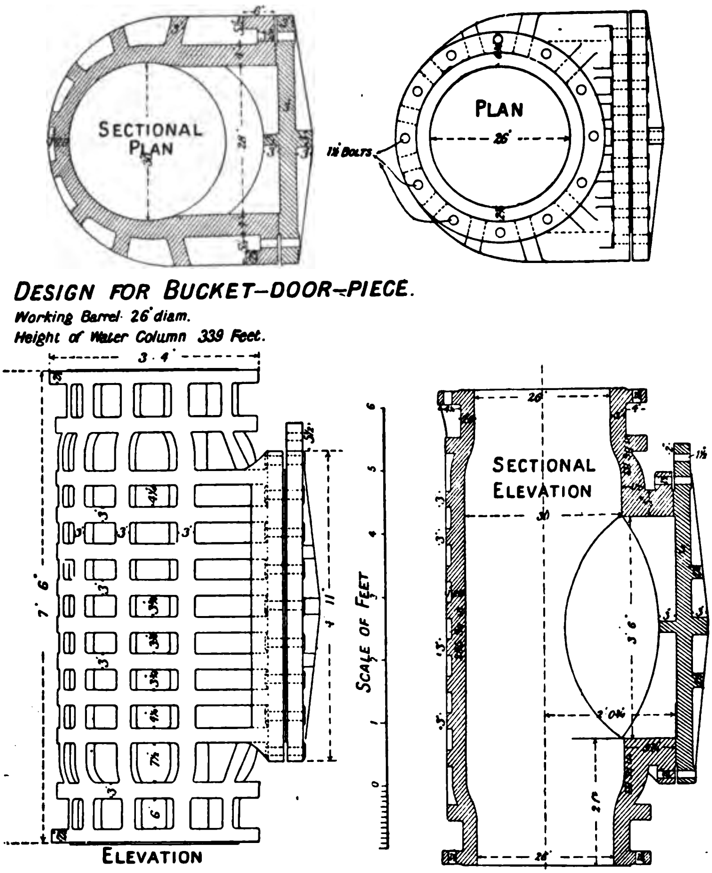


FIG. 571.—Bucket door-piece: design and dimensions.

have daily to go down and to ascend a depth of 1200 feet or more, which is a very serious labour. To reduce this labour a kind of pumping-engine, called a man-engine, has been largely introduced, and is used in the mines of Cornwall, and in many other metalliferous mines of Europe and America; it is shown in Fig. 571*b*. Two rods, like pump-rods, are suspended in the shaft from beams, similar to the pump-beams which are used when pumps are driven by a horizontal engine; the pump-rods work alternately up and down — when one goes up the other goes down, a stroke of say 10 feet. In order to ascend the pit, the workman steps on to a platform, fixed on one of the rods, just

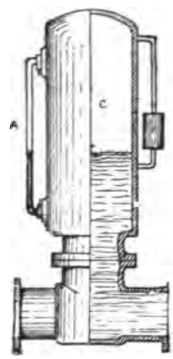
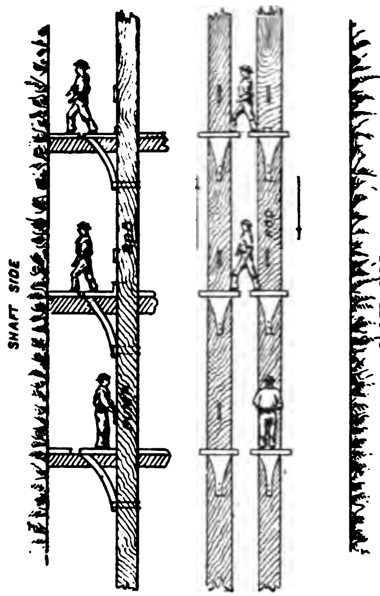


FIG. 571*a*.—Air-vessel. A, glass gauge; B, inspirator; C, compressed air.



Single lift. Double lift.
FIG. 571*b*.—Man-engine.

large enough for one or possibly two men to stand close together. He holds fast by an iron loop fixed to the rod. As the engine at the top is continually moving, the man is no sooner on the platform than he is lifted up 10 feet; at the same time that he is rising, the other rod is coming down. The instant he has got to the top of the stroke, he steps off on to a corresponding platform on the other rod; he is no sooner on, than it goes up 10 feet, whilst the platform he has just left descends. As soon as he has got to the top of the stroke he steps back on to a corresponding platform on the first rod, which is, however, 20 feet higher up than the platform on which he first started. In this way he goes up to the top, first a stroke of 10 feet, then a step off to another platform. If the pump-rods make each six up and down strokes a minute, and he, by changing from one rod to the other, makes a series of upstrokes, he will make twelve upstrokes in a minute—that is to say, he will rise up at the rate of 120 feet a minute, and if the shaft is only 600 feet deep, he will arrive at the top in five minutes; if it is 1500 feet deep, he will arrive at the top in twelve and a half minutes. Sometimes, instead of a double pump, there is only one, and small flat platforms are fixed on the shaft-side on to which the man steps at the top of every stroke, where he waits till the engine has made its downstroke, which brings opposite him a higher platform. With this single pump, it would take him twenty-five minutes to ascend a shaft 1500 feet deep. For descending the pit, he gets on to one of the pump platforms at the top, when it is at the top of its stroke. He is lowered down 10 feet, and steps on to a corresponding platform on the other rod when it is at the top of its stroke; he is again lowered, and so on.

This method of going up and down the shafts is much liked by the miners who are accustomed to it; it is much easier than walking up ladders, and quicker. Practice has made them indifferent to the dangers of missing a step, or forgetting to keep their heads close to the rod when passing through a narrow opening in a fixed platform; whilst the notion of being wound up a shaft 500 yards deep in a minute by a steel rope no thicker than two thumbs, seems to them a dangerous experiment.

CHAPTER XXII.

TREATMENT OF MINERAL ON THE SURFACE: ORE-DRESSING, LEAD, TIN, AND GOLD, COLLIERY SCREENS, COAL-WASHING, COKE-OVEN.

THE chief work on the bank at a colliery is classifying the coal according to size, and loading it into railway waggons, canal-boats, and carts. In addition to classifying, the dirt has to be picked

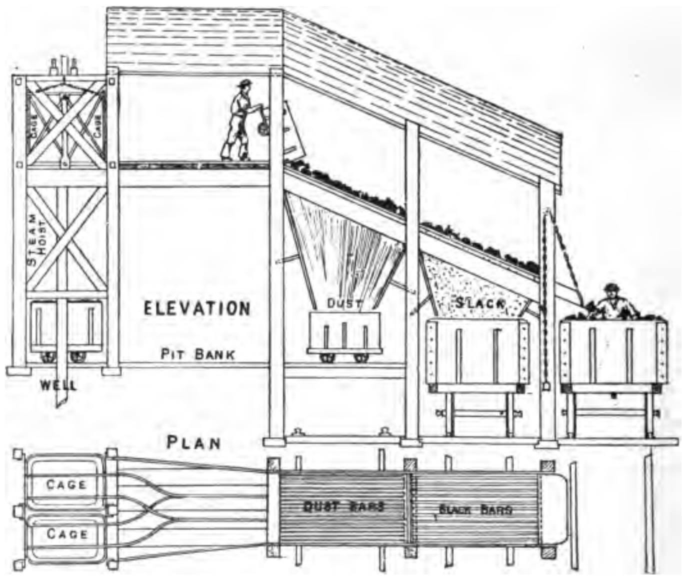


FIG. 572.—Coal-screens and steam-hoist.

out. A plan that is common in the Midland Counties is to run the pit waggon on a tramway, or on plates which are raised about 4 feet from the ground, the railway waggons being close up to this raised bank; the coal is then lifted out of the pit waggons and placed in railway waggons according to size and quality. When there is nothing but small coal left at the bottom of the pit waggon, it is raised by a hydraulic ram, or other hoist, to a higher level, where it is tipped over screens (see Fig. 572), and is made into three or four sizes—cobbles, nuts, peas, and dust. In Yorkshire and the north of England, Lancashire and other

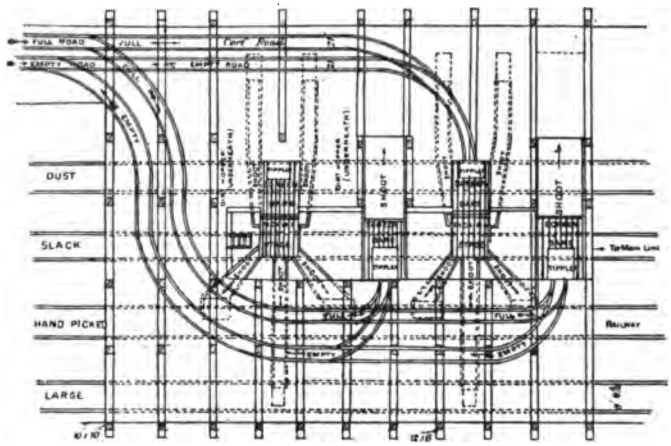


FIG. 573.—Plan of coal-screens.

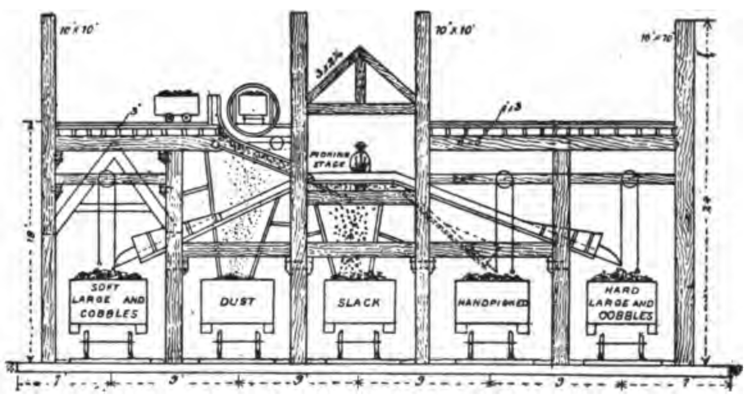


FIG. 574.—Elevation and section of screens shown in plan, Fig. 573.

places, the pit-bank is often raised about 20 feet or more from the ground, and all the coal is tipped down screens (see Figs. 530, 573, and 574), where it is sorted into large, cobbles, nuts, peas, and dust; and also according to quality, such as hard, soft, best, gas, seconds, etc. In order to assist the screening, the screens are shaken by means of a small steam-engine, which gives movement to shaking-rods (suspending the screen bar-frame) by means of small cranks; this causes the coal to slide down shoots of a very moderate inclination, and more effectually separates the dust from the coal. Strong wire network screens are often used for the smaller sizes of coal. Of late years it has become common to tip the coal from the bank down a shoot on to a broad moving band about 4 feet 6 inches wide, made of steel plates 4 feet 6 inches wide by 18 inches long and $\frac{3}{16}$ inch thick, fixed on carrying-chains made of steel bars linked together by pins, so that it goes round rollers at each end; one of these rollers is driven by an engine, and so the band is moved continually forward (see Fig. 575). The coal falling on to this travelling band is taken off by sorters standing at each side, and thrown down side shoots

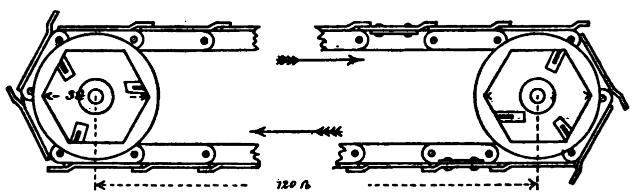


FIG. 575.—Travelling belt.

or direct into railway waggons on each side, according to quality. The dirt is also taken out. The smaller coal falls into a hole at the end of the band, whence it is lifted by a bucket-elevator and thrown over a screen similar to that shown in Fig. 572. The travelling-belt is say 120 feet long on the top, so that any required number of pickers and sorters may stand beside it. Two such belts will deal with 1000 tons of coal in 8 hours. The belts often travel at a speed of 30 to 40 feet a minute.

In the anthracite districts of Glamorganshire and Pennsylvania great care is taken in classifying the coal; the large lumps are often broken up by toothed rollers, so as to make nuts, which are in great demand for stoves. Elaborate and expensive machinery is now erected for the breaking and classifying of anthracite. It is made into all sizes, from large lumps down to the size of peas; the dust is nearly valueless. The broken coal is classified in a manner similar to that employed for metalliferous ores, and shown in Fig. 584; but the crushing-rollers are different, and there is less pressure required to hold the rolls together.

Most bituminous coal can be made into coke. Where there is a great demand for coke, and the coal is qualified to make it of the very best quality (as in the county Durham), the whole production of a colliery is sometimes coked, the large coal being broken up by crushing-rollers to facilitate coking. In other districts only the very small coal, otherwise hardly salable, is coked. The ordinary method of coking, which is considered by many engineers the best, is to put the small coal into an oven built of fire-brick, circular in plan, shown in section, Fig. 576. The coal is put in through a hole at the top; the charge is about 5 tons. If the oven is hot, as it will be if in regular use, the coal takes fire, the side door having been built up with fire-brick and fire-clay. Some small holes are left for the admission of air,

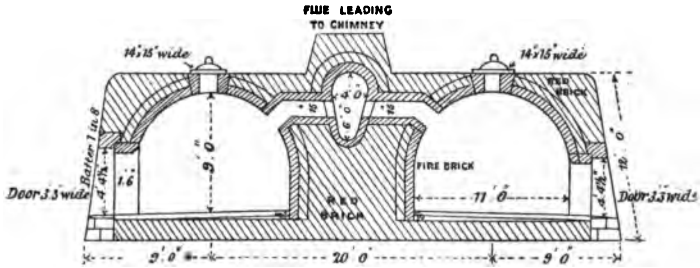


FIG. 576.—Coke-oven.

enough to burn the gases given out by the coal. As the heat of the oven increases by this combustion, the air-holes are stopped by degrees, so that the gases go off unburnt into the flue, which goes to a chimney. It is now usual to introduce boilers between the ovens and the chimney, and to burn the gases under the boilers, thus effecting an important economy. Ovens are generally built in long ranges thirty or forty in a row, and a double row with a central flue leading to the chimney; the outer walls are made very thick, to keep in the heat as much as possible. The intense heat in the oven drives out all the volatile hydro-carbons, and causes the remaining carbon (and dirt) to partially melt and run together into a hard mass, which swells up to a larger bulk than the original coal. This process takes about 72 hours, more or less (it varies very much with the quality of the coal, from 50 to over 100 hours). Then the side door is pulled down, and a water-jet at once turned on to the coke to cool it and so prevent combustion, and then the coke is raked out and further watered (just enough water to stop combustion, but not more than the heat of the coke can evaporate, so that the coke may be dry). The coke is sometimes drawn out at some modern plants by mechanical extractors when it is red hot; the watering is then done outside the oven, and the cooling of the interior by watering is avoided. The oven is immediately recharged with slack. The coke weighs from 50 to 68 per cent. of the weight of the original charge of coal. It breaks up into large irregular lumps, very porous, with bright metallic lustre, grey colour, and specific gravity about 0·74.

Coal-washers.—In case the slack is mixed with a great deal of dirt, it is generally washed. There are several kinds of coal-washers. One of the most largely used kind of washer is constructed as follows: The coal is sized—that is to say, passed through screens which separate it into different sizes (see Fig.

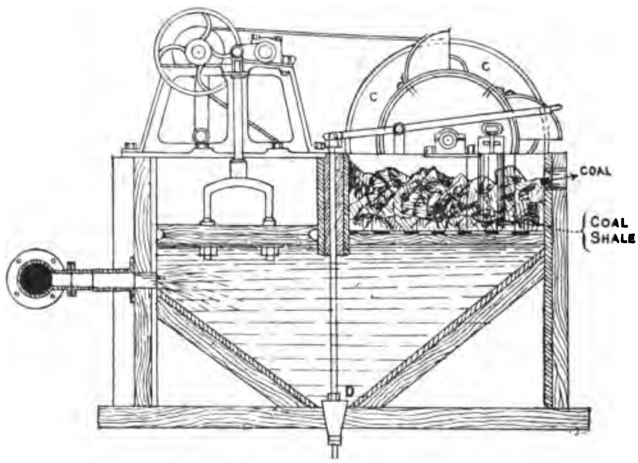


FIG. 577.—Coal-washing "jigger," for coal larger than ⅜" : side view.

584; the rolls shown in the figure are not for colliery plant); each size is conducted to separate washing-machines. The slack is then placed on a tray full of perforated holes (see Figs. 577, 578),[1] in a box which is full of water. By the side of this box is a wooden cylinder, in which there is a wooden piston; this piston is lifted up and down, about fifty or sixty strokes a minute, by a small crank or eccentric driven by an engine; the length of stroke may be varied from 2 inches to 4 inches. The piston makes its downstroke faster than its upstroke. The lifting up and down of this piston causes the water in the box to jump up and down in a corresponding manner; the water, passing through the holes in the perforated tray, lifts up all the fragments of coal and dirt, the lighter fragments (that is, the coal) being lifted the higher.

[1] The reader is referred to an excellent paper on "Coal-washing" in the *Transactions* of the South Wales Institute, by Mr. R. de Soldenhoff, from which Figs. 577–580 have been taken.

As the water recedes the coal and dirt also fall, the heavier particles (that is, the dirt) falling the faster; the result is that in a short time all the dirt is at the bottom and the coal is at the top. The coal is moved off by the water, aided by scrapers, while the dirt goes off up the cylinder A, and out by the opening B, whence it is removed by the revolving scoop C. Some of the small coal and dirt may fall through the gratings, and can be let out by the valve D. This machine will wash about 5 tons of slack an hour.

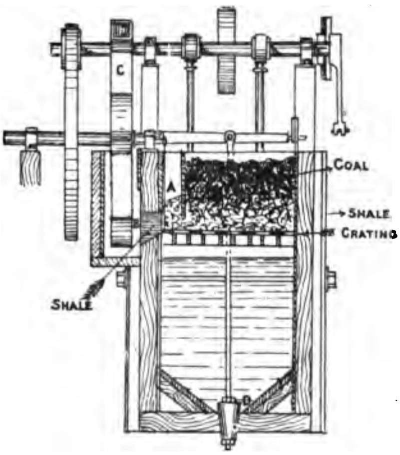


FIG. 578.—End view of coal-washer shown in Fig. 577.

For small sizes of coal, say $\frac{3}{8}$ inch and less, the tray on which the slack is put is covered with a layer of feldspar gravel (Figs. 579, 580) 3 or 4 inches thick. The size varies from $\frac{5}{8}$ inch up to $1\frac{5}{8}$ inch according to size of coal, but is always larger than the openings in the grating. The action is the same as above described. By the action of the water, the dirt sinks down through the gravel and the grating, whilst the coal remains on the top and is carried forward by the water. The dirt, having passed downwards through the feldspar, falls through the perforated tray into the bottom of the box, out of which it is taken from time to time through the opening L. For these smaller sizes of coal the stroke of the piston varies from $\frac{1}{2}$ inch to 2 inches, and from 100 to 150 strokes a minute. This machine will wash from two to three tons an hour.

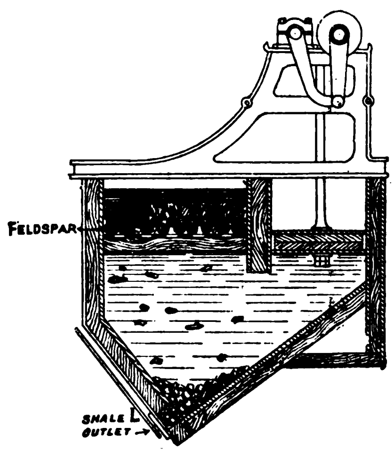


FIG. 579.—Feldspar coal-washing machine for $\frac{3}{8}''$ and smaller: end view

The percentage of coal wasted on this system of washing depends entirely on the careful arrangement of the machine and the outlay incurred. By washing and rewashing the smaller particles, very nearly all the dirt can be extracted, whilst very little coal-slack is lost; but as a rule it does not pay to carry the operation too far. A rough rule at collieries is that ordinary slack containing an average quantity of dirt is reduced in weight one-third in washing; thus 3 tons of dirty slack yield 2 tons of clean slack. Often the percentage of dirt and loss is much less, in some cases 7 tons of dirty slack yielding 6 tons of clean slack.

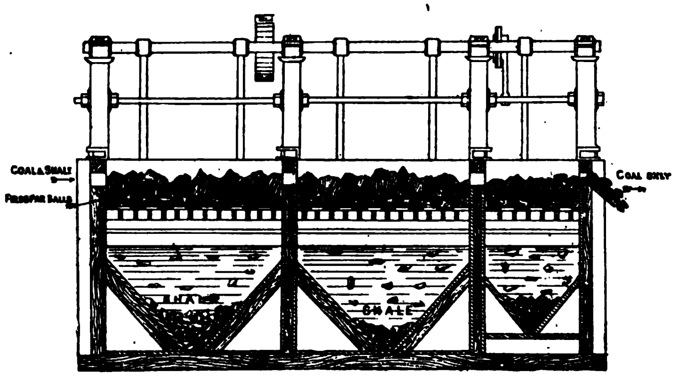


FIG. 580.—Side view of machine shown in Fig. 579.

The water is used over and over again, and there is no occasion for any dirty water to be sent off into the rivers, as has been done in some cases.

It will be seen that the principle of this washer, which is that of most washing and separating machines and contrivances, is to utilize the difference in the specific gravity, or weight for equal sizes, of the coal and shale. The specific gravity of water is 1; that of coal, about 1·3; that of shale, about 2·3; and that of feldspar, about 2·6. Thus the shale, falling faster and rising slower, is separated from the coal; the coal cannot get to the bottom, because each stroke of the piston lifts all the material, and the shale gets down before the coal. The feldspar merely serves the purpose of a grating of great thickness, and in passing through it there is time for particles of coal to be arrested and sent up to the top; it also gives a movable surface to the grating, so that every part of the coal and shale are exposed to the action of the water.

Another kind of machine is Robinson's, which has lately come into use in this country (see Fig. 581). The principle of this coal-washer is the same as that of the jigging-machine, and can be best understood with the aid of the Fig. 581. It consists of a circular iron box, in section like an inverted truncated cone, about 9 feet 6 inches diameter at top, 2 feet at bottom, and 8 feet deep. The dirty coal is filled in at the top, and water is forced in at the bottom. The water is taken from a tank some height above the machine, giving a head of 32 feet, so that it may come at a regular speed and pressure. The water enters all round the

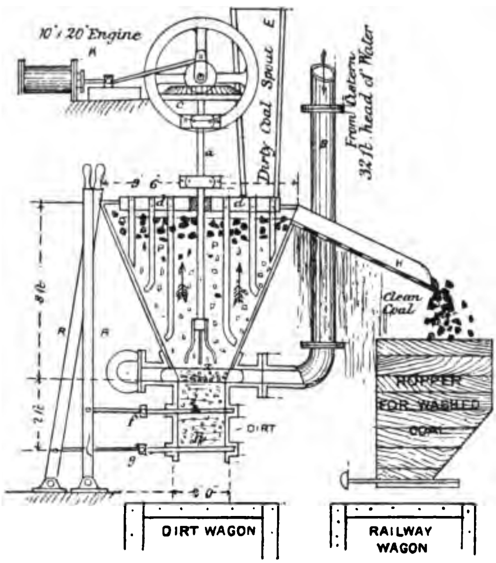


FIG. 581.—Robinson's coal-washer. H, perforated coal-spout; B, supply-pipe to washer; E, dirty coal-spout to washer; P, revolving arms or agitators driven by engine K; R, hand-levers attached to dirt-valves.

cone at the bottom at *x* in the figure, passing through holes about ½ inch in diameter. In order to keep the coal and water mixed, it is stirred up by means of a vertical revolving shaft, *a*, driven by the bevel-wheel *c*, which is driven by the engine K; and on a level with the top of the hopper are four iron arms, *d d*. To each of these four arms are fastened three iron bars, P P. Below the hopper are two valves, *f* and *g*, one of which is always closed to prevent the water escaping at the bottom, so that it is forced to overflow at the top. The space *h* between the two valves forms a receptacle for the dirt, which may be emptied by first closing the valve *f*, then opening the valve *g*. The water passes up at such a velocity that it has power to lift the coal, but has not sufficient power to lift the dirt, owing to its greater specific gravity. Because of the continuous action of this machine, it will get through a great deal of work. As the coal and water come over the side of the hopper it falls down a shoot, H; the water drains off through perforations in the bottom of this shoot.

One of the oldest kinds of coal-washers, which is still extensively used, is the inclined water-trough. From the top of a staging a wooden trough, one or two feet in width, is carried on posts at a sufficient incline for a stream of water to rush with great rapidity; this trough is several hundred feet long. At intervals of a few feet are weirs formed of cross-bars fixed in grooves; the weir is one or two inches in height according to circumstances. The dirty slack is delivered with a stream of water into the trough at the top; the water runs down, carrying with it both dirt and coal; the larger bits of dirt are caught by the wooden cross-bars or weirs, and remain in the trough whilst the coal is carried forward. The deposit of dirt soon fills the trough to the level of the weir. Another cross-bar is then put in the groove; that again catches the dirt, until the space behind is nearly filled with dirt. When this happens the trough is cleaned out, and the process begins again. The water may be pumped up and used over again. This kind of washer gives very good results in some places; it must, however, be borne in mind that the efficiency of any coal-washer depends on the careful classification of the material to be washed.

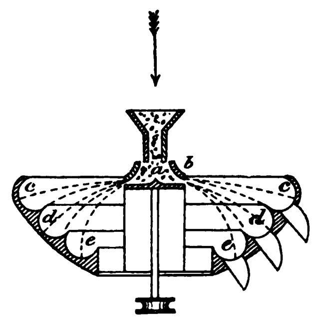


FIG. 582.—Centrifugal ore-separator.

Centrifugal Separator.—One of the drawbacks of the washing of coal is the expense of drying it. If the coal is to be coked, the moisture has to be driven off by the heat of the oven. Sometimes the wet slack is put into a revolving cylinder, so that the moisture can be thrown off by centrifugal force. A dry centrifugal separator has been made for use with metalliferous ores (see Fig. 582).[1] The mixture of ore and dirt is fed into the centre vertical revolving hopper, *a*. This hopper turns with great rapidity, and by the centrifugal force thus produced the stuff is thrown out through the horizontal opening *b*. The heavier material, having greater momentum, is thrown further than the

[1] Clarkson and Stanfield's patent.

lighter, the heaviest furthest of all. Supposing the ore to be separated is lead ore ground to the size of coarse sand, the larger particles will be thrown into the furthest receptacle *c*, the smaller particles into the nearer receptacle *d*, and dirt with very little ore will fall into the receptacle *e*. The dirt and ore mixed can be passed through the separator again.

Briquettes.—A great deal of slack is made into what are called briquettes; these are lumps of slack mixed with some adhesive material and compressed into the form of bricks, whence their name. This is sometimes called patent fuel. Where the coal is exceedingly friable, as at some of the French mines, nearly the whole production of a large colliery is made into briquettes. In England it is only a small proportion of the slack which in some districts is so treated. Many cementing materials have been tried, such as clay and starch, but pitch is now almost exclusively used. The proportion of pitch depends partly on the kind of machinery used, partly on the nature of the coal, and partly on the requirements of the consumer, and may be said to vary between 5 per cent. as a minimum and 20 per cent. as a maximum. The price of pitch is subject to considerable variations, but as its price is not uncommonly ten times that of the slack with which it is mixed, there is every inducement to use it with economy. The process of manufacture is somewhat as follows: The coal is screened and ground to a uniform size; the pitch is ground into a fine powder. The coal and pitch are mixed together in a mill and subjected to heat, generally that of high-pressure steam, which partly melts the pitch; the heated mixture is then lowered into a machine like a brick-making machine, where it is subjected to great pressure, each brick being separately moulded, and, falling on to a travelling-band, is carried away and stacked. This is the plan commonly adopted in English machines. On the continent it is common to force the briquettes out through a long tube, as in a wire-cut brick-machine. But they are forced out by a ram, and not by a pug-mill; the result is that the issuing block of compressed material breaks up into briquettes. Sometimes, instead of square bricks, a cylindrical form is adopted, like that of drain-pipes, but solid inside. When properly made the briquettes are hard, and will stand transport and weathering, and, if made from clean slack, make very good coal, but more smoky than ordinary coal owing to the mixture of pitch. A common weight for English briquettes is about 10 lbs., and they are sometimes made a good deal larger. For domestic purposes this size is somewhat too large, and in France they are made of various shapes and small sizes to suit the householder.

Iron Ore.—The clay-band ore of the coal measures generally requires dressing, because the lumps of ironstone are covered with shale. The practice is to throw iron ore into heaps where it is weathered; sometimes it is watered and washed with a hose pipe, and the shale is afterwards broken off by means of a hammer. The oolite and hematite ores now generally used do not require this dressing. The further treatment of the ore, such as calcining, comes properly under the head of Metallurgy.

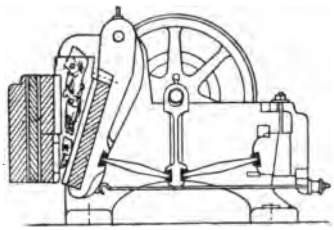


FIG. 583.—Blake's patent stone-breaker.

Lead Ore.—When this ore has been delivered on the bank it is subjected to a process of sorting, large lumps of ore being picked out by hand, and sometimes cleaned by means of a hammer. Lumps of stone and ore mixed are put into a crusher, like Blake's crusher (see Fig. 583); the broken lumps are then put through steel crushing-rolls about 2 feet in diameter, pressed together by a weighted lever (see Fig. 584). In case of some uncrushable material accidentally getting in, the pressure lifts the weight and the rollers part. After passing through the rollers, the crushed stone passes through sizing-screens—that is to say, three revolving screens fastened together in sequence. The first screen has the smallest holes, the second larger holes, and the third the largest. What will not pass through these last is carried back to the rollers. These three sizes of ore fall into wooden troughs, along which the ore is carried by a current of water to jiggers, where it is treated in a manner exactly similar to that already described for the washing of coal. But it must be remembered that in coal-washing the lighter mineral is valuable, and the heavier is dirt; whilst in lead-washing the lighter mineral is dirt, and the heavier is valuable. The dust containing ore is sized in a hydraulic classifier shown in Fig. 585. A stream of water carries the ore and stone into a tank divided into compartments; the heaviest

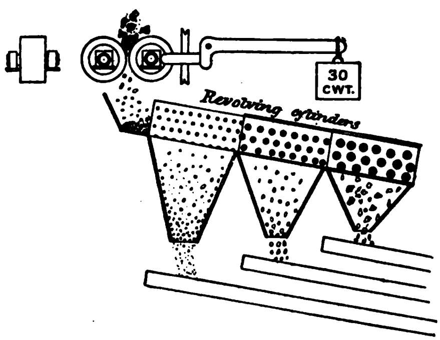


FIG. 584.—Crushing-rolls and sizing-screens.

grains sink into the first compartments, and the lightest of all are carried away by the overflow, and the particles of intermediate size fall into the intermediate compartments, whence they are taken away to the jiggers. The jigging of the water causes the lead to fall down, whilst the dirt is left on the top. The dirt is washed off and carried away with the overflow water, and the lead remains either on the trays or falls into the box below. Instead of the feldspar used in coal-washing, larger-size lumps of lead ore are sometimes put on the trays for lead-ore washing. The overflow water carries away, as well as dirt, fine particles of ore, because the water has no faculty for separating lead from dirt except by the difference in the weight of the lumps, and lead ore in very fine dust is no heavier than coarser dust of dirt.

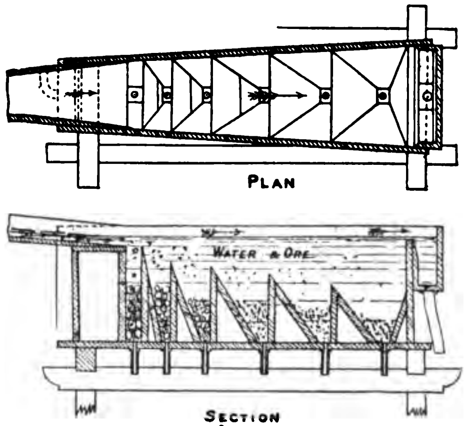


FIG. 585.—Water-compartment classifier.

This slime is, however, washed again by a machine called a buddle (see Fig. 586). This concave buddle is a large basin, say 15 or 20 feet in diameter, the floor of which is made of wood; the sides slope gently towards the centre. Buddles are often convex, the sides sloping down from the centre. In the centre is a vertical spindle made to revolve; arms C C on this spindle move round the basin; dropping from these arms are very soft brushes, which lightly touch the surface of the basin. From a ledge surrounding the basin, fed by revolving pipes or troughs, A A, the slime is delivered by water on to the sides of the basin, and very slowly runs down the sides, the revolving brushes continually stirring it up and turning it round. By this very gentle action the dirt is washed away towards the centre, and some of it is carried away, and the lead remains upon the wood near the outer circumference, from which it is afterwards removed. As the deposit thickens the internal ring B B is raised, and the brushes are lifted a little at the same time by means of the lever D, until the deposit has reached about 1 foot in depth. This deposit is then taken out in three rings: the first or external ring is clean lead ore; the middle ring is ore and dirt requiring rebuddling; the internal ring is mostly dirt.

and may be rebuddled if it contains lead enough to pay. The slope of the buddle is about one in ten.

Tin, Copper, Gold, etc.—The method of dealing with ores above described is applicable to most kinds, and is

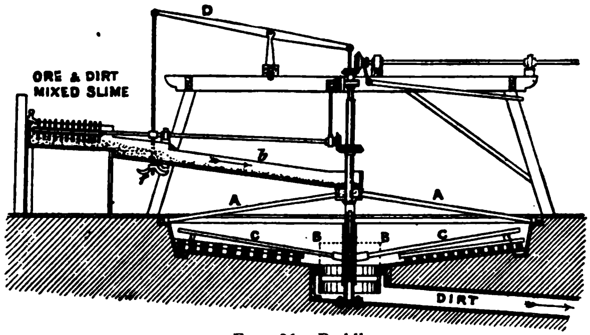


FIG. 586.—Buddle.

largely employed in tin-mines. For tin, however, stamps are generally used to crush the ore to a sufficient fineness. A side view of a stamp is shown in Fig. 587. The stamp has an iron spindle and cast-steel head, weighing together perhaps 7 cwt.; the lower part of the head or hammer is from 2½ inches to 6 inches in diameter according to weight of hammer and hardness of rock. Near the top of the spindle is a collar; a revolving cam raises it and allows it to drop again, the fall being about 8 inches. Below this stamp is a cast-steel or cast-iron anvil; this is in a socket, and can be easily replaced when worn out, and is enclosed in a wooden, iron, or steel box. On one side, the ore, which has been already crushed with rolls, is fed in with a stream of water; at the opposite side is a metal plate perforated with very small holes, through which the ore passes, carried by the water when it has been crushed sufficiently fine. As this process of stamping is very slow, a great number of stamps are required; they are generally arranged in a battery of five stamps (see Fig. 588).

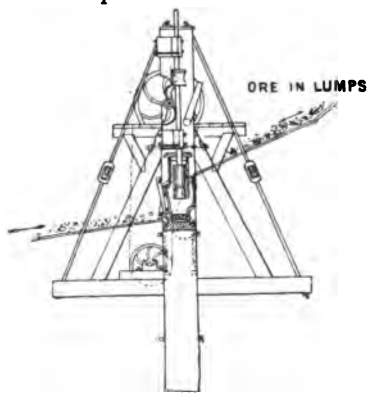


FIG. 587.—Stamp: side view.

The quantity of rock crushed in 24 hours' continuous work depends on its hardness, on the number of blows per minute, and on the size of the openings through which the crushed ore has to pass. As ordinarily used, each stamp will crush from 1 to 3 tons; $1\frac{1}{2}$ to 2 tons is about an average quantity. The number of blows per minute varies from sixty to a hundred. The size of the mesh varies greatly for gold quartz; it is sometimes about nine hundred openings to the square inch; the openings usually vary from $\frac{1}{18}$ to $\frac{1}{60}$ inch in width. The old-fashioned way of working the stamps is shown in Fig. 589. The ore when crushed is taken to the buddle, where the dirt is washed away.

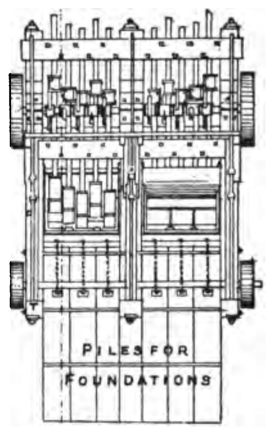


FIG. 588.—Stamp: front view.

Gold quartz is broken up with a crushing-machine on its way to the stamps, where it is stamped exceedingly fine. With the stamps mercury is placed. Mercury has a great affinity for gold, and unites with it, so that a great deal of gold unites with the mercury in the stamp-box. As the slime escapes from the stamp-box through the perforated plates, it is carried by the water over copper plates which have been previously covered with mercury, and the gold in the stream adheres to the copper plates. The residue runs off, and is

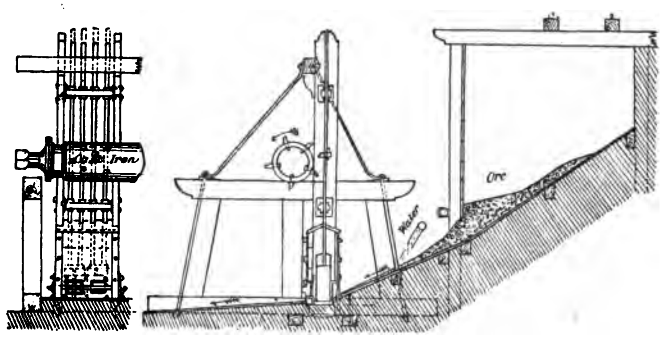


FIG. 589.—Ordinary Cornish stamp.

collected and concentrated. Fig. 590 shows a concentrating-table made of an indiarubber belt about 3 feet 6 inches wide, tightly stretched over rollers about 8 feet apart; there are ledges on each side of the belt. It moves very slowly forward, as shown by the arrows. The slime is poured on from a spout with a stream of water; a very rapid succession of sharp jerks longitudinally from the eccentric throws the heavier ore back, whilst the lighter dirt is carried forward by the water and movement of the belt. The dirt is washed away; the ore is scraped off the upper part of the belt. Another system is like a "Rittinger table." It is similar to the above, but the shaking of the belt and frame holding it is lateral, separating the ore into three streams, all carried forward (see Fig. 591); one is concentrated ore, one is mixed, and one is dirt. In order to extract the gold from the amalgam, it is only necessary to heat the mixture of gold and mercury in an iron retort, when the mercury is turned into gas,

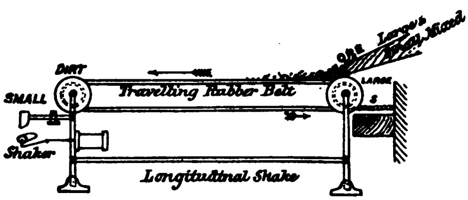


FIG. 590.—Concentrating-table.

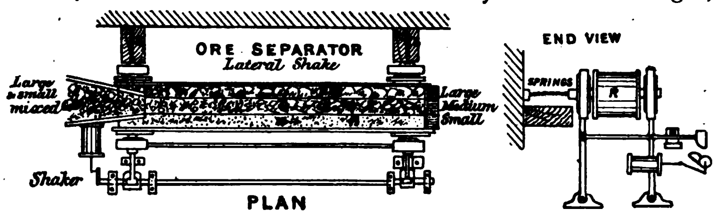


FIG. 591.—Concentrating-table. S, springs; R, travelling rubber belt.

and, going off through a coil, is condensed and can be used over again, while the gold remains in the retort. All gold is not so easily dealt with as this. The above process is applied to what is called "free-milling ore." The chlorination and other processes are used for the extraction of ores mixed with iron and other metals.

In dealing with iron or copper pyrites containing gold, different processes are adopted; a description of these comes under the head of Metallurgy.

In dealing with the diamondiferous rocks extracted from the South African mines, the process is as follows: The earth or rock, technically known as "yellow" or "blue," is laid out on large level floors hundreds of acres in extent. The blue (which is the harder kind of ore, and that which is principally raised) is ploughed and harrowed down to a depth of 9 inches, and is then allowed to weather. In the absence of rain, it is watered by means of pipes. After being sufficiently weathered, it is put into a large trough full of water, where it is agitated by revolving arms. The mud flows off; the heavier parts, including the diamonds, fall to the bottom. It is subsequently treated in jiggers or pulsating-machines, similar to those used for coal-washing, and thus, by a succession of processes, the dirt is separated from the diamonds sufficiently, so that when the concentrated ore is laid out on tables the diamonds can be picked out by hand.

Ore-dressing.—Ore-dressing, or the separating of the metal and the substances with which it may be in chemical combination from the dirt with which it is mixed in the mine, comes within the province of the mining engineer; for instance, lead ore (galena) does not contain pure lead, but sulphuret of lead, which is a mixture of lead and sulphur. It is the business of the ore-dresser to separate the galena from the dirt; it is the business of the smelter to separate the sulphur from the lead. When there is a large proportion of silver, the ore is then called a silver ore, and not a lead ore. Tin is also not found as a pure metal, but as an oxide, and it is this oxide of tin (cassiterite) that the ore-dresser separates from the stone. Copper is sometimes found as pure metal called native copper, as at the mines of Lake Superior, in North America, but it is commonly found as a sulphuret.

The chief modes of ore-dressing resolve themselves into the following simple rules: To crush mineral as it comes from the mine so small that the lumps of pure ore can be separated from the stone, the classifying must take place at every stage of the crushing process; it is done by hand when the lumps of pure ore are not too small, but when the lumps get smaller than the size of a filbert, the sorting is done by machinery, and this is simply done by allowing a stream of water to carry away the lighter or non-metallic particles. But the stream of water must be applied as nearly as possible to particles of stone and ore the same size.

In the case of coal, the lighter particles are the valuable particles, the dirt being heavier; in the case of metals, the lighter particles are the valueless material, the ore being heavier. A shaking movement facilitates the separation of the dirt from the ore, as in the jigger; so in the buddle the revolving brushes give a gentle shake to the ores. This may be attained in other ways, as for instance, a vibrating table or a Rittinger table, an Emery table or a Frue vanner, etc.

In dealing with slimes, it is impossible to separate the ore from the dirt completely at one operation. Some of the dirt is taken away by the first operation, and the concentrated material is then removed to another machine, whether it it be a jigger, a buddle, a Rittinger table, a Frue vanner, or other contrivance, and some more of the dirt is removed, the process of concentration being repeated three or four times, until the ore is sufficiently clean to go to the smelter.

In addition to the processes of concentration where water is used, there are others in which air is used instead of water. In these air-machines air is blown through the ore, blowing away the light worthless particles, whilst the heavy metal falls down. This is simply the old-fashioned process exemplified in the winnowing-machine, by which chaff is blown away from wheat; but the application of the principle to ore-dressing involves a great difference in construction. Machines have been made to separate ore without any air-blast, in which the crushed ore is ejected from a rapidly revolving centrifugal machine (see *ante*, p. 468).

There is also a great variety of crushing-machinery. This crushing is sometimes done by rollers revolving in a pan, as in a mortar-mill. In other machines large balls of cast iron are placed in a pan containing ore; the pan is made to revolve, and the ore is crushed by the balls rolling over the ore.

In other crushing-machines the ore is whirled round with great rapidity by revolving paddles, which throw the ore against the internal projections of a strong cast-iron case; another set of revolving paddles by the side of the first set moves in the other direction. In another machine the ore is injected by an air-blast through a nozzle. A similar air-blast, exactly opposite to the first one, injects an opposing stream of ore, the opposing volleys of ore being thus broken up.

Crushing-machines may be divided into three classes: first, that in which the ore is crushed between a hammer and anvil; second, a roller and pan; third, that in which one piece of ore is made to strike against another piece of ore. Most of the crushers belong to the first and second classes.

In many places there is not sufficient quantity of ore or sufficient capital for the erection of powerful machinery, and a great part of the work is done by hand and shovel, or by means of a jigger worked by hand.

Water-power is largely used for driving mining-machinery, especially in mountainous districts; overshot wheels and turbines, as well as other wheels, are used, a 40-feet overshot water-wheel being capable of driving fifteen stamps for gold quartz, together with concentrating-machinery and all other apparatus necessary for extracting gold from the ore, equal to dealing with 30 tons of hard quartz per diem.

In considering the modes of dealing with valuable minerals, it is necessary to make some mention of slate. Perhaps no other material is so wastefully treated, doubtless on account of its abundance. Out of the whole produce of a slate-quarry only from 5 to 10 per cent. of the slate is marketable; the rest is unmarketable, chiefly so rendered by the process of mining or subsequent dressing. For economy of mining it is necessary to blast the slate, and this causes the greater portion to be shattered so as to be worthless. It is necessary to prepare the slate for market immediately it is got, because if it has time to dry it cannot be split up. It is frequently got in large lumps, as shown in Fig. 592, from a photograph by the writer

FIG. 592.—Lump of slate.

These lumps are conveyed on rails into the workshop, and lifted by cranes on to tables connected with circular saws; by means of these saws they are cut into the required shape, whether the slate is wanted in the form of large slabs or to be split up for roofing slates. Fig. 593 is from another photograph taken by the writer,

FIG. 593.—Splitting slates.

showing a workman in the act of splitting a slate with a chisel driven into the edge by a hand-hammer. Having driven the chisel in a little way, he then opens the crack already made by a movement of his left hand, holding the chisel, and then he takes both hands to tear the slates in two. Beside him is a man cutting the slates square and the edges straight. He does this by means of a heavy knife hinged at one end and held up by a spiral spring; by means of a treadle, he brings the knife down by the side of a fixed iron straight-edge, on which he holds the slate which is to be cut.

CHAPTER XXIII.

MISCELLANEOUS PRECAUTIONS FOR SAFETY; DISCIPLINE; BLOWERS OF GAS, OUTBURSTS OF COAL; DAMS OF WOOD, BRICK, ETC.

THE question of safety, as is inevitable in any treatise on mining, has been continually mentioned in the preceding chapters, but it may be well to refer more particularly to certain points. The Mines Regulations Act of 1887 contains a great deal of instruction which it is unnecessary to recapitulate in this treatise, as it should be obtained separately by every student of mining. The student must get the Act itself; not the "abstract," which is misleading in some parts. The Act is supplemented in each district by special rules, which must also be studied by the student, to see if they contain additional instructions.

The table on the following page, extracted from the Government Returns, shows the proportions of fatalities under various headings.

It will be observed that the falls of roof and stone are the most frequent causes of accidents. The danger from this cause is chiefly to be met by careful timbering. When a collier is holing, he must put sprags under the coal. But there is also a liability of coal bursting off from the face; he must, therefore, put sprags against the face of the coal, as shown in Fig. 423. To protect himself against falls of the roof, he must put up timber in the manner described in the chapter on "Timbering."

It is necessary constantly to examine, not only the roof, but the sides of every road, because, owing to the joints in the shales, clays, coal, and stone, masses weighing hundredweights or tons are constantly falling off, and, if they fell on a passing man or boy, might inflict serious or fatal injuries. An experienced workman passing along a road, examining by sight and by touch, can ascertain if any ground is loose, and if it is he can at once pull it down or secure it with timber. This has to be done before any workman enters a place, and from time to time during the working-hours; in fact, it is necessary to have every place under continual observation.

TABLE XXIII.—NUMBER OF ACCIDENTAL DEATHS.

Causes.		Coal Mines (Coal Mines Regulation Act).							Metalliferous Mines (Metalliferous Mines Act).						
		1887.	1888.	1889.	1890.	1891.	Total for 5 years.	Average per ann.	1887.	1888.	1889.	1890.	1891.	Total for 5 years.	Average per ann.
Explosions of fire-damp		149	49	138	290	51	677	135·4	5	1	—	—	—	6	1·2
Falls in mine	Falls of sides ...	106	125	115	100	119	565	113·0	23	25	28	14	24	114	22·8
	Falls of roof ...	364	346	350	334	357	1751	383·6							
In shafts	Overwinding	8	1	2	1	9	21	4·2	1	—	—	—	—	1	0·2
	Ropes and chains breaking ...	1	1	2	3	1	8	1·6	2	—	—	2	—	4	0·8
	Whilst ascending and descending by machinery ...	16	17	22	20	23	98	18·4	2	2	4	3	2	13	2·6
	Falling into shaft	6	5	8	7	7	33	6·6	1	3	—	—	1	5	1·0
	Things falling from surface ...	3	3	1	5	5	17	3·6	—	1	—	—	—	1	0·2
	Falling from part way down	21	30	17	20	44	132	26·4	4	6	5	4	7	26	5·2
	Things falling from part way down	6	6	5	8	5	30	6·0	—	2	—	1	2	5	1·0
	Miscellaneous	23	12	17	24	25	101	20·2	3	—	3	—	1	7	1·4
Miscellaneous underground	Explosions of gunpowder ...	22	15	23	29	25	114	22·8	7	9	6	6	1	29	5·8
	Suffocation by gases	6	8	9	11	5	39	7·8	—	3	—	—	—	3	0·6
	Irruptions of water	3	3	4	3	1	14	2·8	—	—	—	3	2	5	0·1
	Falling into water	—	3	—	1	—	4	0·8	—	—	—	—	—	—	—
	On inclined and engine planes	65	59	75	77	80	356	71·2	2	—	1	3	1	7	1·4
	Trams and tubs	73	81	80	87	80	401	80·2	—	—	1	—	..	1	0·2
	Machinery underground ...	10	9	6	3	7	35	7·0	1	2	2	—	—	5	1·0
	Sundries underground ...	34	31	95	34	38	232	46·4	4	11	5	2	5	27	5·4
On surface	Machinery on surface ...	6	14	10	21	10	61	12·2	—	2	3	2	2	9	1·8
	Boilers bursting	4	1	5	3	2	15	3·0	—	—	—	3	—	3	0·6
	Railway sidings at the mine	69	29	43	43	44	228	51·6	1	5	6	3	3	18	3·6
	Miscellaneous	—	40	37	36	41	154	30·8	—	—	—	—	—	—	—
Gross total		995	888	1064	1160	979	5086	1017·0	56	72	64	46	51	289	57·8
Tonnage raised, coal, fireclay, shale, and ironstone, under the Coal Mines Act		173,049,795	182,660,163	189,633,656	194,605,887	197,693,592	937,643,093		—	—	about 3,000,500	4,151,310	about 3,990,000	—	—
Work-people employed		526,277	534,945	563,735	613,233	648,450	—	—	41,749	43,472	43,420	42,054	39,428	—	—
Persons employed per life lost ...		529	602	530	528	662	—	—	745	603	678	914	773	—	—
Tons of mineral wrought per life lost		173,919	205,698	178,227	167,763	201,934	—	184,357	—	—	54,687	90,246	78,235	—	—

If any evidence of this were required, it can be obtained by going into some air-road where the workmen do not pass, and wnere repairs have not recently taken place. This road will very likely be heaped with masses of fallen material, showing the observer what has recently occurred, and what is likely to happen again ere long. Sometimes the efforts to secure the roof by packs and timbers are unavailable, and what is called a "weight" comes on. This happens when the adjoining pillars of coal are too distant for the strata to bridge over the space, and then the whole of the strata up to the surface have to be supported by the props and packs, which, being necessarily insufficient for this purpose, give way. This weight generally gives warning; sounds of cracking and bumping are heard, and the workmen have time to escape.

When timber props have to be withdrawn, great care is necessary, as mentioned in the chapter on "Timbering."

Old Workings.—It is an everyday occurrence for some mine to be closed, and the place so abandoned will probably, so much of it as is not closed by the overlying strata sinking down, in time become filled either with noxious gas or with water. Any miner working a new mine in the vicinity of such abandoned mine, must be guided by an accurate plan of the old mine and of the new mine, showing their relation one to the other. The method of so producing an accurate plan is not a subject for this book, but is dealt with in a separate treatise.

It frequently happens, however, that one or both plans are inaccurate or incomplete (indeed, it seldom happens that a colliery plan, either of a modern or ancient mine, is perfectly accurate), and the miner, through this cause or through inadvertence, may cut a hole into the old workings, and gas or water may issue from these with disastrous consequences. In order to prevent such an accident, the following precautions are taken :—

When approaching near the abandoned mine, the old shafts are cleared out, and the water pumped out of the abandoned mine. Where this is done the danger of a serious catastrophe is very slight. It has happened, however, in some cases that workings not shown on the old plan, below the level drained by the pumping-engine, have remained full of water, and an accident has resulted. Therefore, in addition to any other precautions, it is necessary, and required by law, that any working place supposed to be approaching old hollows should not be more than 8 feet wide, and should be protected by sufficient front and flank bore-holes. These bore-holes are small holes, say not exceeding 2 inches in diameter, carried in advance of the heading.

Fig. 594 shows a heading (6 feet wide) with front and flank bore-holes. The front bore-holes are usually kept from 10 to 20 yards in advance of the heading, and this is easily done if the bore-hole can be kept parallel to the inclination and direction of the heading, because as each yard of heading is cut it is only necessary to advance the bore-hole in front an equal distance. The flank bore-holes should be started from the face of the heading, and put in a distance of 9 feet at an angle of say 26½° from the direction of the heading, which is equal to a divergence of 1 foot for every 2 feet in the line of heading. After the heading has been advanced 6 feet, the first bore-hole must be deepened to a total length of 18 feet, or even up to 30 feet, measuring from the origin at the face of the heading. A flank hole in each side must be started every 6 feet; each flank hole should start 1 foot 6 inches from the centre of the heading, as shown in the figure. The length of the bore-holes must be regulated by the nature of

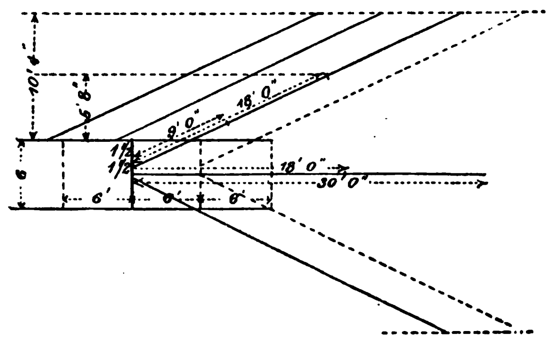


FIG. 594.—Plan of heading approaching old hollows, showing bore-holes.

the ground and the pressure of water expected, and the promixity of the flank bore-holes by the size of the headings in the old workings. Sometimes these are narrow, and the bore-holes may miss them unless they are close together. Where very great pressures of water are expected, the bore-holes should be put in much longer than in the above example.

The writer has found great advantage in substituting for solid iron boring-rods hollow rods of steam-pipe; these are much lighter and stiffer. The part of the rod outside the hole is carried on a roller; a cross-head is fixed to the end of the rod, and several workmen operate it.

The work of boring is tedious and expensive with men who are unpractised, but after a few weeks' practice it becomes comparatively rapid and cheap. Where the seam is very thick, it is necessary to have several holes, one above the other, both in front and in flank; because the old workings may be in a higher or lower portion of the seam, which must be treated as divided into separate beds each about five feet thick, and borings must be made in each bed. If these precautions are not merely ordered, but carried out, the chances of driving into old workings unexpectedly are reduced to a minimum.

Plugs should also be at hand in the heading, with which to stop up the bore-hole in case gas or water should issue into the working place; and the workmen, as well as the officials, must examine every symptom of increase of water or issuing gas, so that they may not be taken unawares. The pressure of water may sometimes burst away a considerable thickness of coal, and so cause a deluge.

Blowers of Coal.—Some mines are subject to outbursts of coal and gas. These are very dangerous, because the miners are liable to be smothered by the coal. These outbursts of coal come from the face of the bank or heading, and a portion of the seam is blown out from the face, perhaps as much as 20 tons being suddenly ejected from the face of the coal, chiefly in the form of dust, but sometimes there are small pieces. The issue of dust is generally accompanied by an issue of fire-damp. An interesting account of the outbursts at the Broad Oak Colliery, near Ashton-under-Lyne, has been given by Mr. Joseph Dickinson, F.G.S., and published in the *Proceedings of the Manchester Geological Society*, part 8, vol. 22, 1893. At this colliery the section was as follows: Inferior coal and dirt, 6 inches; impure soft coal and sulphur combinations, 1 foot 6 inches; hard coal, 1 foot 10 inches. The holing is done in the soft coal, and the blowers that take place are blowers of soft coal only. The coal seems to burst or leap out—a kind of spontaneous and instantaneous holing or curving—sometimes to a depth of nearly 8 yards from the face, and to a width of nearly 8 yards. The cavity thus spontaneously holed is irregular, often wider at the back than at the face. These outbursts are generally preceded by cracking, and accompanied by a rumbling noise, as of thunder or steam blowing from a boiler. Outbursts of a similar kind have occurred in the Black Vein in South Wales, and in Belgium.

Opinions differ as to the cause. The more probable explanation, however, seems to be that, owing to the depth of the mine and the softness of the seam, it is crushed so as to lose its natural form and cohesive nature, and that it is then projected by the expansion of the gas in the coal. There are difficulties in accounting for the ejection of the coal by the gas, because in some cases the blowers have occurred where previous workings would, in ordinary course, have drained away most of the gas, and numerous bore-holes made in the working places have failed to let off the gas so as to prevent the blowers. It may, however, be assumed that the coal is different in its nature in some parts of the mine, and that in the places where the blowers occur it is rather softer than elsewhere; so that a very small pressure of gas, insufficient to break down even a very thin rib of strong coal, is still sufficient to clear out what is simply a heap of coal-dust in a cavity protected by the harder coal above and below and all around.

Blowers of Gas.—Precautions have to be taken, not only against the everyday dangers of the mine, but against unusual accidents, such as the issue of large blowers of gas, against which any amount of ventilation is useless. Should such blowers occur, the only safety lies in the absence of any exposed light, a defective safety-lamp, or a furnace.

A case is recorded in Belgium of fire-damp issuing from the mine so strongly that it fired at a light in the winding-engine house on the surface, and then blazed in a column 50 yards high over the pit-top, and continued to blaze for hours, the pit being so full of gas that there was not air enough in the shaft for a flame or explosion. As the blower of gas became exhausted, the mixture of air caused explosions to take place in the shaft, and ultimately in the mine below. Issues of gas on a minor scale have frequently occurred in the coal-mines of this country, and in fiery districts may be constantly expected.

Safety-lamps.—A great many of the coal-mines of the country are worked with naked lights, such as candles or oil-lamps, and the metal-mines are always so worked. In some places, parts of the mine are lighted with gas, which is forced down the shaft by a species of pump or by falling water or steam. The gas is only used for lighting the porches adjoining the pit-bottom. Electricity is now being substituted for gas with advantage. Safety-lamps are used in the majority of coal-mines, even though gas is hardly ever seen, and then only in small quantities. The reason for using safety-lamps in these cases is threefold. One is to avoid the danger in case of a blower; the second is to avoid the small accidents which are likely to occur from the workman lighting with his candle a small accumulation of fire-damp, the injury from which will very likely be not fatal, but cause a good deal of alarm, or because such a small explosion of fire-damp might in some cases cause a catastrophe through the ignition of coal-dust; the third is that the use of naked lights has sometimes caused fires by the firing of brattice-cloth, or timber, or other inflammable material, due, of course, to the gross neglect of some person.

In some respects it is a pity to use safety-lamps where it is not necessary, because they give such a poor light as compared with candles, and a good light helps the workman to see that his place is secure from falls of roof and side. It also greatly helps his labour, and therefore gives him more time and energy to spare for the due timbering of his place. It also saves him from injury to the eye resulting from the use of insufficient light. Dr. Court, of Staveley, has shown that a large percentage of miners using safety-lamps are subject to a disease of the eye called miner's nystagmus.[1] The remedy is to obtain a portable safety-lamp that shall give as good a light as a candle; this, however, has yet to be invented. In the mean time, it may be said that two or three good lamps would be much better than one, and as the extra expense would be trifling, there is no doubt that the coal-cutter might have a much better light than is usual if he strenuously desired to protect his eyesight from injury.

Duplicate Roads.—There should also be two roads—shafts or other passages available for every workman—so that in case one way is stopped by some accident, he has another road for escape. This is applicable to every mine, metalliferous as well as coal.

Shaft-sides.—The sides of the shafts must be carefully examined to see that the walling, timbering, or bare rock is sound, and all machinery in good order. Indeed, it is essential that every part of the mine, above ground and under ground, where the workmen go, and every part of the machinery, should be examined at least once a day. Full instructions for this and for the reports, consequent on the examination, are contained in the Mines Regulations Act, and the special rules. The reader is cautioned against the abstracts of the Act, published by the authority of the Government, and posted up at the mines, because these abstracts are, in some important respects, misleading, and not in accordance with the Act itself; and it is the Act, and not the abstract, that the courts of law enforce.

Discipline.—It is essential to the safety of a mine that there should be efficient discipline. It is useless for the manager to give orders unless they are carried out. As a rule, the workmen

[1] Mr. Simeon Snell, ophthalmic surgeon to the Sheffield General Infirmary, has published an interesting book showing that nystagmus is found amongst miners working with candles, and is of opinion, in which he appears to be supported by medical men on the continent, that this disease is due to the position in which miners have to work, and that it is not caused by the use of safety-lamps. After carefully reading this valuable book and Dr. Court's able paper, the writer is of opinion that the balance of evidence shows a much greater prevalence of miner's nystagmus where safety-lamps are used than where candles are used.

obey directions which they see are necessary for their own protection, and even rules the need for which they do not understand, if they see that the manager is in earnest.

It is more important that punishment for breach of rules should be certain than that it should be severe; indeed, severity is seldom necessary.

Since, in a large mine, it is impossible for the manager to go over it every day, and sometimes the manager has a larger extent of mines under his care than he can find time to see oftener than once a month, the safety of the mine depends on the deputy-managers—or deputies, as they are called for short—who visit every place two or three times during the twenty-four hours. These officers should have a complete understanding of everything that is requisite for the safety of the mine, and a responsibility equal to their understanding for the preservation of the lives of the workmen. Where that is done accidents are rare, and great accidents never happen except where some important truth has not been fully realized; as, for instance, the fact that coal-dust is an explosive agent, and the chief agent in destructive colliery explosions. Before this had been, as it is now, definitely established, no one could be blamed for ignorance of a fact apparently so contrary to daily experience; but now, when it is known, that knowledge will probably tend in the future, as it has already tended in the past six years in a very marked degree, to diminish the risk of colliery explosions.

Every coal-mine, according to the Act of 1887, is under the daily care and supervision of a manager, and every mine, where more than thirty men and boys are engaged underground, must be under the care of a manager holding a first-class certificate. If from any cause the manager is absent from the mine, another manager must take his place. This manager may have a second-class certificate, and is called in the Act the under-manager.

At large collieries it is now the practice to have a manager and under-manager, one of whom is at the works every day. These managers are represented in every district of the mine by deputies. The duties of the manager and under-manager are far in excess of anything that any one man can perform; the result is that nearly the whole of the daily routine work of supervision is performed by the deputies.

Each district has a deputy, and there is a separate deputy for each shift, so that a mine divided into four districts and working three shifts would have at least twelve deputies. Every working place and travelling-road has to be carefully examined by a deputy, and a written report made of the examination before the workmen enter the mine or the district of the mine.

There are sometimes deputies specially appointed to look after the haulage—that is to say, to supervise the hurriers, drivers, engine-men, and others engaged in the conveyance of mineral underground; and other deputies are specially responsible for the repairs of the main air-ways and renewals of timber in the main roads. There is also a head or senior deputy for the whole mine, who has charge in the absence of the manager and deputy-manager from the pit, and a senior deputy in each shift. In fact, the organization is as complete as that of a well-officered army, and in a large well-managed colliery all the work should go on correctly in the absence of the manager and under-manager. All the machinery and engine-men are under the charge of a mechanical engineer, called an engine-wright; he has also deputies or assistants to take his place in his absence. Blacksmiths and carpenters are at the works, and generally a bricklayer. The horses and ponies are under the charge of a horse-keeper and his assistants, and the stores are under the charge of a store-keeper and his assistants.

The safe and economical working of a colliery depends on having a capable and responsible official, always ready and always present in every district, to prevent or remedy an accident.

Safety-masks.—In entering a mine containing noxious gas, say one where there has been an explosion of coal-dust, it is often in the highest degree desirable to be able to walk through the noxious gas as a diver walks through water. Suppose, for instance, as happened in a recent case, that a disastrous explosion occurred, killing a large proportion of the men in the pit, and breaking down some of the stoppings and overcasts by which the ventilation was guided. Then the part of the mine beyond the broken stopping and overcast would be unventilated, and the after-damp, which is so poisonous, would perhaps remain in the place where it was formed; and districts of the mine that had escaped the explosion might retain an atmosphere that could be breathed, and the men who were working there might remain safely until a roadway was made for their release. If, now, the ventilation were restored whilst the men were in the mine, the after-damp would be blown upon them, and might poison them before they could be reached by the rescue-party. Also the explosion might have set fire to some timber or coal, which fire would be partly extinguished by the after-damp; but if fresh air were introduced, the fire might blaze up.

If it were possible to walk through the roads full of after-damp, and explore the mine before restoring the ventilation, these dangers might be to some extent obviated. With this object in view, many inventors have contrived apparatus to enable a man to walk through foul air. It is evident that, if a diver can walk along the bottom of the sea, he might, in the same costume, walk through a poisonous atmosphere. But the costume of a diver is unsuited for crawling about a mine, and the long pipe that the diver has attached to him, and by which he is supplied with air, is also unsuitable for dragging through roadways of a mine immediately after an explosion.

Perhaps, amongst the numerous inventions, the most convenient is that known as the Fleuss apparatus (see Fig. 595). This consists of a mask for the mouth and nose, with indiarubber pads tightly pressed against the face, so that the wearer of this mask cannot breathe in any air except through the opening provided. Behind him, on his shoulders, he carries a small pack, which contains a small cylinder of compressed oxygen, and a box containing tow and caustic stick-soda. There are two small pipes connecting the mask with this pack; one of these pipes conveys the air that he exhales, or breathes out, into the pack. There is a valve in this pipe which only opens one way, so that the wearer can only exhale through it, and not inhale, or suck in, air through the pipe; so that he cannot breathe over and over again the air that he has exhaled until it has been into the pack. The other pipe conveys revivified air from the pack to the mask. This pipe also has a valve opening only one way, so that air can be sucked from the pack along it, and not blown back.

FIG. 595.—Fleuss patent noxious gas apparatus.

The exhaled air is purified by passing by the caustic soda, which absorbs the carbonic acid gas, and it is revivified by an admixture of oxygen from the cylinder. A small tap in the cylinder permits a regulated supply to escape to mix with the air. Thus the same nitrogen is continually passing in and out of the lungs of the man, but a fresh supply of oxygen is taken out of the cylinder. (The oxygen is stored in the cylinder at a pressure of 40 atmospheres.) The time that the man can wear this mask depends on the supply of oxygen and the rate at which he uses it. It is generally made so that the supply of oxygen may last for more than one hour. This apparatus has been practically tested upon several occasions, and has been successful. It is, however, difficult to induce a man to put on such a contrivance to which he is unaccustomed, when he has to crawl and scramble with difficulty over heaps of fallen stone.

In order that the mines may have light, it is necessary that he should have a lamp also that will burn in gas, whether carbonic acid or fire-damp.

A lamp was made by Mr. Fleuss, giving a brilliant light, and supplied by oxygen from a cylinder. It is entirely independent of the atmosphere of the mine, the products of combustion bubbling out through water. Whilst this lamp gives a fine light, it requires a large supply of oxygen, and it is probable that a portable electric lamp—a species of safety-lamp not invented when the Fleuss apparatus was first brought out—would be better.

It is seldom that the use of this apparatus is heard of, because every colliery manager believes that there will never be an explosion at *his* colliery, and deems it unnecessary to go to the expense of providing an apparatus for such a remote contingency. It is probable that the only way in which the readiness of such an apparatus can be ensured, would be by a law enabling the County Council to provide the apparatus at suitable stations all over the mining districts of the country, with miners trained to its use, and practised from time to time. In this way there would be sufficient and efficient men, and sufficient and efficient apparatus, always ready for use—like lifeboats and their crews—in any emergency, if the circumstances rendered it applicable.

Oxygen gas compressed into cylinders has been found useful in assisting recovery from the effects of after-damp.

After an Explosion.—It may be useful to give a few suggestions as to the method usually pursued in exploring a colliery after a disastrous explosion. Fortunately, with the improvement of safety-lamps, with the understanding of the dangers of coal-dust, and with the introduction of improved explosives and general enlightenment of the mining community, there has been of late years a great reduction in the number of disastrous explosions; especially is it so in comparison with the increased tonnage of coal, and therefore many colliery managers of great practical experience have had few or no opportunities of assisting in the exploration of a mine immediately after an explosion.

The writer has had some little experience, having assisted in the exploration of five important collieries after a disastrous explosion, and therefore makes the following suggestions, which are founded, not simply on his own experience, but on the practice of others with a much larger experience.

Assuming that the colliery manager is on the surface when the explosion takes place, his first practical step is to get the winding-machinery into working order, and to clear one of the shafts, so that he and others may be able to descend and ascend.

If the mine is ventilated with a furnace, it is usual to extinguish this at the first opportunity

An immediate search is made about the pit-bottom for survivors of the explosion, and, if there are any, inquiries are made from them as to the nature of the accident.

The explorers will proceed with safety-lamps cautiously, on account of the dangers of after-damp, which is very deadly, and therefore they must not proceed in a body further than the ventilation may extend; but the advance must be cautious, one or two men leading, and others following up by twos some yards behind, to render assistance if necessary, in case those leading should be unexpectedly overcome by poisonous gas.

The return air-roads must be immediately examined for smoke or any other signs of fire that may exist in the mine, as well as for fire-damp and after-damp.

If the stoppings and overcasts are blown down, it will be necessary to repair these temporarily with timber and cloth, in order to clear away the after-damp, so as to permit of an exploration of the mine. The danger to be apprehended is that there may be a fire in some part of the mine, and that the restoration of the ventilation may drive some accumulated fire-damp over this fire, and thus cause another explosion—a catastrophe which has happened more than once. To avoid this, the ventilation must be restored gradually, the explorers going forward as quickly as possible to see if any fire exists in the mine, and, if the after-damp permits them, in advance of the ventilation.

The amount of risk that may be taken in the exploration of the mine depends upon the probability of finding any survivors. If there is such a probability, a considerable risk is sometimes rightly incurred; if there is not such a probability, it is not right to incur much risk. No men should be allowed in the mine except those absolutely necessary for the exploration until the mine has been traversed throughout, and it has been established that no fire exists. When that has been done, there is no reason why as many workmen as can be usefully employed should not be permitted to enter the mine. There must, however, be a constant look-out maintained in the return air-roads for all signs of fire, as well as for foul air.

When the stoppings, doors, and overcasts of a mine have been blown down, the ventilating currents are very uncertain, and are subject to reversals, and after-damp may in this way be suddenly blown upon an exploring party with fatal results. It is, therefore, as a general rule, advisable to restore the ventilation as quickly as possible. If there are no falls of roof blocking the road, this can be done very quickly by means of brattice cloth and timber. It usually happens, however, that there are falls of roof which block the road and impede the work. If there is no way round these falls, a road sufficiently large for men to pass through has to be made over the fallen stones.

For the purpose of exploration in foul air, the Fleuss apparatus, previously mentioned, has been found useful in some cases.

Temporary Ventilator.—In case the mine has been previously ventilated by a furnace, or in case the mechanical ventilator has been too much injured for use, steam is sometimes used to produce a current by taking it down the upcast shaft in a pipe, turning it upwards so as to form a jet. The mistake usually made with these steam-pipes is in not taking them sufficiently far down the shaft. The result is that only a very short column of hot air exists at the top of the shaft, and the ventilation produced by this short column is very uncertain, and liable to reversal, which is a *source of great danger.* If the steam-pipe were let down the shaft for a hundred yards or more, there would then be a long column of hot air, and there would be no danger of the ventilation being reversed by a change of wind or other cause.

Dams.—In the chapter on "Sinking," the tubbing of the shafts is described as a means of keeping the shaft dry in passing through watery strata. It sometimes happens that a horizontal shaft or gallery passes through watery strata, and that it is necessary to make it dry in a similar manner. This is sometimes done by lining the tunnel with brickwork set in cement in the form of a barrel arch, and sometimes with cast-iron tubbing in a similar manner to that adopted in vertical shafts, which will be readily understood without further description.

In other cases the workings of a mine proceed into some district where a good deal of water is met with, say by infiltration from some river or lake, or bed of rock communicating with the outcrop. In some cases where the mine is under the sea, the vein or seam of coal has been pursued incautiously until it nearly approached the bottom of the sea, which has caused an enormous influx of water. In many of these cases it is necessary to put a dam into the gallery or heading leading into the district where the water is met with. In case the district is reached by one or two narrow headings, it is not very difficult or very expensive to put in these dams. One method of doing this is shown in Fig. 596, which gives a plan and section of a heading with a dam of brickwork. In this country bricks and cement can generally be obtained in a few hours, and they therefore form the most convenient mode of making a dam.

A nick, the width of the intended wall or dam, is cut into the sides of the heading for a distance of from 1 foot 6 inches to 3 feet, according to the nature of the ground; if in coal, say 3 feet.

The nick is also cut up into the roof and down into the floor, to a height and depth of from 18 inches to 3 feet, according to the hardness and firmness of the strata. A wall is then built across the heading, filling up the nicks. If the pressure is not great, the wall may be straight; but if there is a high pressure, the wall must be built in the form of an arch, as shown in the

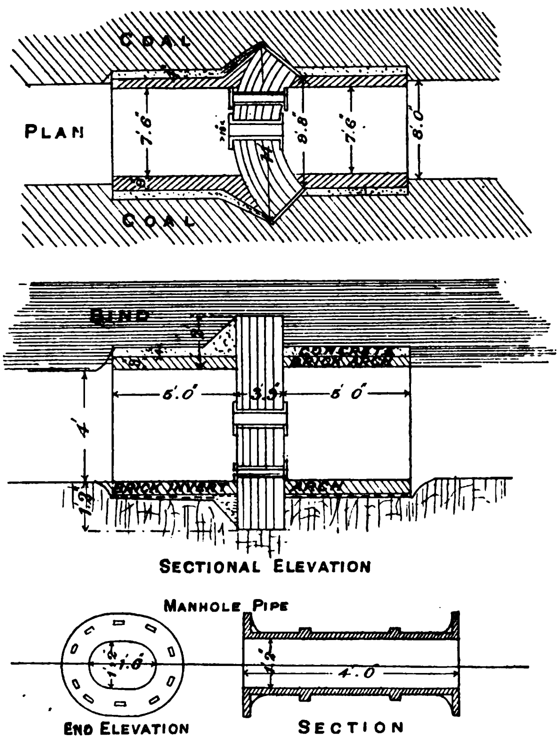


FIG. 596.—Brick dam.

plan, so that it cannot be broken except by the crushing of the bricks; the convex side of the dam being, of course, turned to meet the pressure. Since, of course, the strength of the wall can be no more than that of its component parts, it is necessary that the hardest bricks should be used. The wider the arch, the more important it is to have hard bricks. The cement must also be of the best quality, and, to resist a high pressure, should be used without admixture of sand. The bricks must be laid in separate rings, as in building a bridge, and between each ring a space must be left filled in with pure cement. Where the wall fits against the coal or stone, pure cement must be poured in to fill up any interstices, and to run into the cracks, if any, in the strata. Each ring of brickwork must be built up to the top as tightly as possible, and plastered with cement at its junction with the natural strata, so as to leave no opening for the water to pass through. In a wall made of eight or nine rings of bricks, these separate plasterings should be sufficient to prevent the water permeating over the top. In order to prevent the roof from breaking down, or the floor from heaving up on either side of the dam, a brick arch may be put in for a length of 4 feet on each side, backed with concrete. This arch may be put in after the dam has been built.

As it is necessary for workmen to work at the back of the dam during its construction, a man-hole has to be provided. This is generally done by building in a cast-iron pipe, say 18-inch by 14-inch bore, on to the front flange of which a cover is bolted.

There is also another pipe below the man-hole to take the feeder of water. This lower pipe will, therefore, vary from say 6 inches up to 12 inches in diameter. This pipe will be closed by a cover bolted on. A pressure-gauge may be fitted on to this cover, so as to indicate the pressure that comes against the dam.

In some cases the dams are made of wood in the following manner. Wooden blocks are cut to the length of the intended thickness of the dam, say 3 or 5 feet. They are rectangular in section, and tapered, so as to be narrower in front than at the back, and when laid side by side form an arch. The ground is cut out as shown in the figure for a brick dam. The bottom is carefully levelled, and the wooden blocks laid in the form of an arch. Where they join the natural strata, wedging is put in to make a water-tight joint. The wall is then built up, layer upon layer, and upon the top of the last layer wedging is driven in to make a water-tight joint. A man-hole is left, and closed at last by drawing into the hole a tapered key-piece, and a water-pipe is built in as in the case of a brick dam. The joints of the pieces of wood may be caulked at the back and front. The wood should be put in quite dry; it will then swell when it becomes wet, and make a water-tight joint. Nothing can be better than a wooden dam if carefully made.

To calculate the pressure of water tending to crush such a dam as is shown in Fig. 596, the following rule may be adopted: *Multiply the radius of the outer convex ring of bricks in feet by the pressure of water per square foot; the product is the total pressure on each abutment for each foot in height of dam.*

To find the pressure on each square foot of abutment, divide the total pressure, as given by the above rule, by the thickness of the dam in feet.

Example.—Thus in Fig. 596, the radius of the outer ring of bricks is 10 feet 6 inches; the depth of water pressing on the dam is 62 yards; this equals a pressure of 80 lbs. per square inch, or 11,520 lbs. per square foot. Then, applying the rule, $11{,}520 \times 10 \cdot 5 =$ pressure in pounds on each abutment for each foot in height of dam. The dam is 4 feet thick, therefore the pressure on each square foot of the abutment of the dam equals $\frac{11{,}520 \times 10 \cdot 5}{4} =$ 30,240 lbs.; or, in other words, the pressure of the water is $13\frac{1}{2}$ tons per square foot of abutment.

CHAPTER XXIV.

MINERS' TOOLS: SHARPENING, TEMPERING, ETC.

THE tools used by miners consist for the most part of the pick, crowbar, shovel, riddle, wedge and hammer, drill, rammers, and scrapers. The pick is commonly made of cast steel, and sometimes of shear steel. When made of shear steel, the body of the pick is made of iron, and the steel welded on; when made of cast steel, the whole piece is steel.

The shovel is generally made of cast steel, but iron with a steel edge is sometimes used.

The drill is commonly made of cast steel, all in one piece, and sometimes the end only is shear steel forged on to an iron bar.

The wedges are generally made of iron, sometimes of steel.

The hammers are often made of cast steel.

A pick-point is generally sharpened by a blacksmith, but miners are frequently able to sharpen their own tools. To sharpen a pick, it has to be heated sufficiently for forging; it is then put on the anvil, and a fine point given to it by careful hammering. This heating and slow cooling whilst it is hammered softens the steel, and to fit it for use it must be hardened again. For this it has to be raised to a cherry red or glowing red, and then suddenly dipped into cold water for a depth of say $1\frac{1}{2}$ inches measuring from the point, and moved about in the water so as to ensure sufficient cooling for a period of from 3 to 5 seconds. The effect of this sudden cooling from a cherry red is to harden the pick, but to make it too brittle. The end of the pick is now quickly rubbed on a stone, on which is some clean fine sand; this cleans the point. Dipping the point in the water for this short time does not cool the pick all through, and the heat remaining in the body of the pick re-heats the point, and unless this reheating is checked it will make the point too soft. By watching the cleaned surface of the steel a change of colour will be perceived; a light straw colour will first appear, and shortly after, a deep blue. These colours appear first at some distance from the point, say one inch, and the blue appears like a ring chasing the straw colour to the point. The pick must now be dipped again in cold water, where it may remain until cold. Sometimes the pick is dipped when the straw colour is at the point, sometimes it is dipped just as the blue reaches the point; this latter is the time often used for coal picks. If the pick is dipped too soon, the point will be hard, but brittle; if too late, the point will be tough, but soft. If the proper temper is missed, then the three processes of heating, hardening, and tempering must be repeated. The sharpening of the pick has thus three processes: first, the forging of the point; second, the re-hardening of the steel; third, the tempering of the steel.

Drills are sharpened, first, by forging to the right shape and to give a sharp edge; this edge, however, by many smiths is not hammered sufficiently sharp, and they use either a file or a grindstone to give the required edge. The point is then heated to a glowing red and dipped in cold water for a few seconds to harden the steel; the edge is then rubbed on sand to clean it. The smith examines for the colour, and dips at a pale straw colour to make it hard, or at a dark blue, which makes it a little tougher. If, after the first cooling, there is not sufficient heat in the drill for these colours to show on the edge, it must be reheated in the fire. When the drill is dipped for tempering, it may remain in the water till cold. The exact colour at which steel has to be dipped varies with the quality of the steel, and also, no doubt, with the nature of the work, but a little practice will soon show.

In twisted drills for cutting coal or stone, the cutting-edge has to receive an angle suited to the kind of material in which it is to bore. Thus, supposing the steel were cut across at right angles to the length of the drill, it would have no cutting edge, and would not screw into the mineral; if, on the other hand, the steel is cut to the edge of a chisel in the direction of the length of the drill, the drill could not be turned round, and the edge would be liable to be broken. Therefore the corner or cutting edge of the steel must be at an angle say of 45° with the direction of the length of the drill; but this angle has to be altered as experience shows for each mineral. Thus in a hard coal the edge of the cutter might make an angle of 50° with the direction of the length of the drill, whereas in a softer coal an angle of 40° might be suitable. This edge is partly forged and partly filed; it is then hardened and tempered in the same way as if it were a pick.

Miners' picks were formerly each one provided with a wooden shaft, which was fixed in the pick by a wedge. It is now a common plan to cover the end of the shaft with steel, over which the eye of the blade can be dropped from the handle-end downwards. The steel-covered pick-handle being slightly tapered and largest at the bottom, the pick blade is jammed on. In some cases the pick-blade is put through a steel slot fixed on to the end of the shaft and fastened by a wedge. The advantage of having the blade separate from the shaft is that the miner can carry half a dozen picks in his pocket, whereas if shafts have to be carried as well, several picks are very cumbersome.

Formerly it was customary in many collieries, as well as in other mines, to have a small blacksmith's shop in the pit for sharpening tools. Modern practice, however, is against all fires in coal-mines, and the sharpening-shops are generally at the surface. In cases where miners sharpen their own picks, heating-furnaces are provided with a number of small openings through a brick wall; the miner can just put the point of his pick through the opening into the furnace. In this way facility is given for a number of miners to sharpen their picks at the same time.

Pick-shafts may be made from ash; American hickory is much liked, split into suitable sizes. They are often shaped by machinery, on lathes of suitable construction.

APPENDIX.

Rules for calculating the Size of Steam-engines, with Examples.—The following rules and examples are not intended to guide the engineer in fixing the exact size of an engine required for the specified work, but merely to give rough approximations. Almost any rule that can be devised will be incorrect in some cases. The examples given are not in any way intended as recommendations, but merely to show the student how to work out a calculation, and, when he has had some practice in this, he will be able to apply his knowledge to the more careful and accurate working out of the dimensions of any machinery which he may have to design.

Winding Engines.—*Two Cylinders coupled.*—The load of the tubs and coal, cage, chains and rope, on the drum in tons × 10 = breaking-strain of the rope.

Circumference in inches of the rope (best plough steel) (p. 423)—

$$= \sqrt{\frac{\text{breaking-strain in tons}}{4}}$$

Make diameter of drum in feet = $0{\cdot}7C^2$

C = circumference of rope in inches.

(*Memo.*—This rule is not founded on any scientific theory, but merely represents good practice. If plough-steel ropes are used, the diameter of drum, as calculated by this rule, is less than general practice.)

Size of Engine—

a = area of one cylinder (of the two) in square inches.
d = diam. " " " in inches.
s = stroke in inches.

Area of a circle = $0{\cdot}7854d^2$

Make $s = 2d$.

Piston-travel in inches for one revolution = $2s = 2 \times 2d = 4d$

The unbalanced load of coals (and rope if unbalanced) on the drum in pounds } × circumference of drum in feet
= foot-pounds per revolution of drum.

Add $\frac{1}{4}$ or multiply by $1\frac{1}{4}$ to cover friction of machinery.

Then $0{\cdot}7854d^2 \times \text{steam pressure} \times \frac{4d}{12}$ = foot-pounds per revolution of engine for each cylinder = $\frac{0{\cdot}7854d^3 \times \text{steam pressure} \times 4}{12}$

Make foot-pounds per revolution of engine for each cylinder equal foot-pounds per revolution of drum × $1\frac{1}{4}$. Then by substitution—

$$\frac{\text{Foot-pounds per revolution of drum} \times 1\frac{1}{4} \times 12}{0{\cdot}7854 \times \text{steam pressure} \times 4} = d^3$$

The stroke = $2d$

Make the engine on the other side of the drum the same size.

This rule makes the engine powerful enough to lift double the actual load with twice the actual friction. The margin of power is required for acceleration in ordinary working, and to permit of dirt-tubs or other heavy weights being raised at a slower speed.

Example.—Depth of shaft, 500 yards. Average effective steam pressure on piston = 45 lbs. per square inch.

Load on the Rope—

4 tubs @ 5 cwt.	= 1 ton
4 loads of coal @ 10 cwt.	= 2 „
Cage and chains, say	= $1\frac{1}{2}$ „
Rope 4 inches circum., say	= 2 „
	$6\frac{1}{2}$ tons

Size of rope of best plough steel. Factor of safety = 10.

$$6{\cdot}5 \text{ tons} \times 10 = 65 \text{ tons breaking-strain}$$

By the rule given on p. 423, a rope of best plough steel has a breaking-strain = $4C^2$.

$$65 \text{ tons} = 4C^2$$

$$\therefore \sqrt{\frac{65}{4}} = C, \text{ or (say) } \sqrt{16} = 4 \text{ inches} = \text{circum. rope.}$$

Weight of rope by rule, p. 423 : C^2 = pounds per fathom ; $C^2 = 16$; 500 yards = 250 fathoms ; 16 × 250 = 4000 lbs., or say 2 tons.

$$\text{Diameter of drum} = 0{\cdot}7C^2 = 0{\cdot}7 \times 4^2 = 0{\cdot}7 \times 16 = 11{\cdot}2 \text{ feet}$$

Working diameter of drum = 11·2 feet + diameter of rope = 11·31 feet.

The tubs and cage ascending balance those descending.

The unbalanced load on the drum =

4 loads of coal @ 10 cwt.	= 2 tons
500 yards 4-inch rope	= 2 tons
	4 tons = 8960 lbs.

lbs. Circum. of drum.

8960 × 11·31 × 3·14 = 318169·6 foot-pounds per revolution of drum

Add $\frac{1}{4}$ = 79542·4 to cover friction of machinery

397712·0 = foot-pounds per revolution of engine

$$\text{Then } \frac{0{\cdot}7854d^3 \times 45 \text{ lbs.} \times 4}{12} = 397712 \text{ foot-pounds}$$

$$\therefore \frac{397712 \times 12}{0{\cdot}7854 \times 45 \times 4} = d^3 = 33759$$

$\sqrt[3]{33759} = 32·32$, or, allowing for sectional area of piston-rod, say 30 square inches, $d =$ (say) 33 inches
Stroke = 33 inches × 2 = 66 inches = 5 feet 6 inches

Pair of engines, each 33-inch cylinder, 5 feet 6 inches stroke, 11·2-feet drum.

The diameter of drum, as calculated by the rule above given, is less than general practice, according to which the drum would be about 18 feet in diameter. Adopting this larger drum, the calculation would be as follows:—

lbs. Circum. of drum.
8960 × 18 × 3·14 = 506419·2 foot-pounds per revolution of drum
Add $\frac{1}{4}$ = 126604·8 to cover friction of machinery

633024·0 = foot-pounds per revolution of engine

Then $\frac{0·7854 d^3 \times 45 \text{ lbs.} \times 4}{12} = 633024$ foot-pounds per revolution

$$\therefore \frac{633024 \times 12}{0·7854 \times 45 \times 4} = d^3 = 53733$$

$\sqrt[3]{53733} = 37·73$, say 38 inch cylinder

Stroke = $2d$ = 38 inches × 2 = 76 inches = 6 feet 4 inches

Make the engine on other side of drum same size.

Revolutions of engine per journey $= \frac{500 \text{ yards} \times 3 \text{ feet}}{18 \text{ feet} \times 3·14} = 26·5.$

Pair of engines, each 38 inches diameter, 6 feet 4 inches stroke, 18-feet drum.

Coupled Winding Engines.—11 2-*feet drum, rope balanced.*

Ft.-lbs. per rev. of engine for each cylinder, unbalanced rope = 397712
2

" " " for both cylinders = 795424
Deduct foot-pounds of rope 4480 × 11·31 × 3·14 = 159084

Foot-pounds for the two cylinders of balanced engine = 636340
Foot-pounds for one cylinder $= \frac{636340}{2} = 318170$

Then by rule—

$$\frac{318170 \times 12}{0·7854 \times 45 \times 4} = d^3 = 27007$$

$\sqrt[3]{27007} = 30·03$, say (allowing for piston-rod) 31-inch cylinders
31 × 2 = 62 inches = 5 feet 2 inches stroke

Pair of cylinders 31 inches diameter, 5 feet 2 inches stroke.

Coupled Winding Engines.—18-feet drum, rope balanced.

Ft.-lbs. per rev. for each cyl. unbalanced rope = 633024
2

" " " for both cyls. = 1266048
Deduct ft.-lbs. of rope 4480 × 18 × 3·14 = 253209

1012839 = ft.-lbs. for 2 cyls.

$$\frac{1012839}{2} = 506419 = \text{ft.-lbs. for 1 cyl.}$$

Then by rule—

$$\frac{506419 \times 12}{0{\cdot}7854 \times 45 \times 4} = 42986$$

$$\sqrt[3]{42986} = 35{\cdot}03$$

Stroke = 35 × 2 = 70 inches = 5 feet 10 inches.

Two cylinders 35 inches diameter, 5 feet 10 inches stroke.

Main and Tail Rope Hauling Engine.—To design an engine to haul a given tonnage per hour up a plane of given length and gradient at a given speed with a given steam pressure at engine. If the weight of each tub and its capacity are not given, the student must assume these himself.

If the gradient varies, the steepest gradient must be taken.

A. $\frac{\text{Length of plane in feet}}{\text{speed of set per min.}}$ = time in minutes to travel one way.

B. Time to travel one way × 2 = time to travel in and out.

C. Time in and out + an allowance for changing at each end = total time per journey.

$$\frac{\text{1 hour}}{\text{total time per journey (C)}} = \text{number of journeys per hour} = D$$

$$\frac{\text{Tonnage required per hour}}{\text{No. of journeys per hour (D)}} = \text{tonnage per journey} = E$$

$$\frac{\text{Tonnage per journey (E)} \times 20}{\text{capacity of one tub in cwts.}} = \text{number of tubs in the set} = F$$

1. Gravity due to set

$$= \frac{\text{weight of tubs and coal}}{\text{denominator of fraction representing gradient}}$$

2. Friction of set

$$= \frac{\text{weight of tubs and coal}}{\text{denominator of fraction representing friction (say } \frac{1}{50})}$$

3. Friction of ropes $= \frac{\text{weight of ropes}}{\text{denominator of friction fraction } (\frac{1}{12})}$

(1, 2, 3 together:) (G) Total load on the drum

In calculating the gravity and friction due to the rope, assume as nearly as possible the size of the rope, which has a breaking-strain of not less than 6 times the load.

By rule on p. 423, C^2 = lbs. weight per fathom ;

Then $\frac{2 \text{ length of plane in feet} \times C^2}{6}$ = assumed weight of rope in lbs.

Actual Size of Rope required.—

$$\frac{\text{Total load on drum in lbs.}}{2240} \times \text{factor of safety (6)} = \text{breaking-strain of the rope in tons}$$

If it is decided to have a crucible steel rope, by rule on p 423, $3C^2$ = breaking-strain of rope in tons.

$$\therefore \sqrt{\frac{\text{breaking-strain in tons}}{3}} = \text{the circumference of rope required}$$

Size of Engine required.—Assume the diameter of the drum.

a = area of cylinder in square inches.
d = diameter of cylinder in inches.
s = stroke in inches.

Make $s = 2d$.

Piston-travel in inches for one revolution $= 2s = 2 \times 2d = 4d$.

$$\frac{4d}{12} = \text{piston-travel per revolution in feet}$$

$$\left\{\begin{array}{c}\text{The load on the}\\ \text{drum in pounds}\end{array}\right\} \times \left\{\begin{array}{c}\text{circumference of the}\\ \text{drum in feet}\end{array}\right\} = \left\{\begin{array}{c}\text{foot-pounds per revo-}\\ \text{lution of drum}\end{array}\right.$$

To this add 50 per cent. (*i.e.* multiply by $1\frac{1}{2}$) for contingencies, such as extra friction on curves and rollers, extra weight such as dirt, friction of engine and drums, fall in steam pressure, etc.

Single Direct-acting Engine.—

$$\text{Revolutions of drum per minute} = \frac{\text{speed of rope per minute feet}}{\text{circumference of drum in feet}}$$

Foot-pounds in engine per revolution $= 0{\cdot}7854d^2 \times$ steam pressure

$$\times \frac{4d}{12} = \text{foot-pounds per revolution of drum} + 50 \text{ per cent.}$$

$$= \frac{0{\cdot}7854d^3 \times \text{steam pressure} \times 4}{12}$$

$$\therefore \frac{\text{foot-pounds per revolution of drum} \times 1\frac{1}{2} \times 12}{0{\cdot}7854 \times \text{steam pressure} \times 4} = d^3 \text{ (Rule)}$$

$\sqrt[3]{d^3}$ = diameter of cylinder in inches. The stroke $= 2d$.

Coupled Direct-acting Engine.—

$$\frac{\text{Foot-pounds per revolution of drum} \times 1\frac{1}{2} \times 12}{0{\cdot}7854 \times \text{steam pressure} \times 4 \times 2} = d^3$$

Single-geared Engine.—

$$\frac{\text{Foot-pounds per revolution of drum} \times 1\frac{1}{2} \times 12}{0{\cdot}7854 \times \text{steam pressure} \times 4 \times \text{gearing}} = d^3$$

Coupled-geared Engine.—

$$\frac{\text{Foot-pounds per revolution of drum} \times 1\frac{1}{2} \times 12}{0{\cdot}7854 \times \text{steam pressure} \times 2 \times 4 \times \text{gearing}} = d^3$$

Examples.—*Main and Tail Rope Hauling Engine.*—Tonnage per hour, 60. Length of plane, 2000 yards. Steepest gradient against the load, 1 in 10. Weight of full tub, 15 cwt. Weight of empty tub, 5 cwt. Average effective steam pressure on the piston

45 lbs. Average speed of set = 6 miles per hour. Time taken in changing at each end, say ½ minute.

$\frac{5 \text{ miles} \times 1760 \text{ yards} \times 3 \text{ feet}}{60 \text{ minutes}} = 528$ feet average speed of set per minute = periphery speed of drum per minute

$\frac{2000 \times 3}{528} = 11{\cdot}36$ minutes time to travel one way (A)

$11{\cdot}36 \times 2 = 22{\cdot}72$ minutes time to travel in and out (B)

$22{\cdot}72 + 0{\cdot}5 = 23{\cdot}22$, say 24 minutes total time taken per journey (C)

$\frac{60}{24} = 2\frac{1}{2}$ journeys per hour (D)

$\frac{60 \text{ tons}}{2\frac{1}{2}} = 24$ tons per journey (E)

$15 \text{ cwt.} - 5 \text{ cwt.} = 10$ cwt. capacity of tub

$\frac{24 \text{ tons} \times 20}{10 \text{ cwt.}} = 48$ tubs in the set (F)

48 tubs @ 5 cwt. = 12 tons
tonnage of coal = 24 tons

36 tons weight of tubs and coal

$2240 \times 36 \text{ tons} = 80640$ lbs.

Gravity of set $= \frac{80640}{10} = 8064$

Friction of set $= \frac{80640}{50} = 1613$

Friction of main and tail ropes $= \frac{18000}{12} = 1500$

11177 lbs., total load on drum (G)

Friction of Ropes.—

$\frac{2000 \text{ yards} \times 2}{2} = 2000$ fathoms of rope

Assume a 3-inch circumference rope = 9 lbs. per fathom

$2000 \times 9 = 18000$ lbs. of rope

$\frac{18000}{12} = 1500$ lbs. friction of ropes

Actual Size of Rope required.—Factor of safety = 6.

$\frac{G}{2240} = \frac{11177}{2240} = 5$ tons

$5 \times 6 = 30$ tons breaking-strain

By rule on p. 423, a crucible steel rope has a breaking-strain = $3C^2$.

$3C^2 = 30$

$\therefore C = \sqrt{\frac{30}{3}} = \sqrt{10} = 3{\cdot}16$, say 3¼-inch rope

Size of Single Direct-acting Engine.—Say drum = 6 feet diameter, say 19 feet circumference.

11177 lbs. × 19 feet = 212363 foot-pounds per revolution of drum
Add 50 per cent. = 106181 for contingencies

318544 foot-pounds per revolution of engine

By rule given—

$$\frac{318544 \text{ foot-pounds} \times 12}{0{\cdot}7854 \times 45 \text{ lbs.} \times 4} = d^3 = 27038$$

$\sqrt[3]{27038} = 30{\cdot}02$, say 30 inches diameter of cylinder
stroke = 30 inches × 2 = 60 inches = 5 feet

$\frac{528 \text{ feet}}{19} = 27{\cdot}8$ revolutions of engine or drum per minute (average), say 28

Coupled Direct-acting Engines.—

$$\frac{318544 \text{ foot-pounds} \times 12}{0{\cdot}7854 \times 45 \text{ lbs.} \times 4 \times 2} = d^3 = 13519$$

$\sqrt[3]{13519} = 23{\cdot}82$, say 24-inch cylinder
Stroke = 24 inches × 2 = 48 inches = 4 feet

Pair of engines 24 inches diameter, 4 feet stroke.
Average revolutions of engine or drum per minute = 28.

Single-geared Engine.—2 to 1.

$$\frac{318544 \text{ foot-pounds} \times 12}{0{\cdot}7854 \times 45 \text{ lbs.} \times 4 \times 2} = d^3 = 13519$$

$\sqrt[3]{13519} = 23{\cdot}82$, say 24-inch cylinder
Stroke = 24 inches × 2 = 48 inches = 4 feet
Average revolutions of drum per minute = 28
" " engine " = 28 × 2 = 56

Coupled-geared Engine.—2½ to 1.

$$\frac{318544 \times 12}{0{\cdot}7854 \times 45 \text{ lbs.} \times 4 \times 2 \times 2\tfrac{1}{2}} = d^3 = 5407$$

$\sqrt[3]{5407} = 17{\cdot}55$, say 18-inch cylinders
Stroke = 18 inches × 2 = 36 inches = 3 feet
Average revolutions of drum per minute = 28
" " engine " = 28 × 2½ = 70

N.B.—This is an unusually high speed for constant work.

Endless Rope.—Refer to method of working out the total load on the engine in the Main and Tail system. If the gradient varies, take the *average gradient.*

$$\frac{\text{Delivery required in tons per hour} \times \text{length of plane in feet}}{\text{speed of rope per hour in feet}} = \text{tons of coal on the rope}$$

$$\frac{\text{Tonnage on the rope} \times 20}{\text{capacity of 1 tub in cwts.}} = \text{number of full tubs on the rope}$$

There will also be the same number of empties on the rope. The tubs and rope coming outbye balance the tubs and rope going inbye.

Gravity of *coal only*, on the rope }
Friction of *full tubs and empties* ($\frac{1}{50}$) } total load on the driving-wheel
Friction of rope ($\frac{1}{12}$) }

Length of rope = 2 length of plane

Assume the size of the rope, and work out its weight as in the Main and Tail system.

$$\frac{\text{Speed of rope in feet per minute} \times \text{load on the engine in lbs.}}{\text{piston speed per minute}} = \text{total}$$

pressure required upon the piston

Assume the piston speed at 250 feet per minute.

$$\frac{\text{Total pressure on piston}}{\text{steam pressure}} = \text{theoretical area of cylinder}$$

Add $\frac{1}{2}$ for friction of machinery, etc.

Stroke = diameter of cylinder × 2

$$\text{Revolutions of engine} = \frac{\text{piston speed}}{2\ \text{stroke}}$$

If we assume size of driving-wheel, then—

$$\frac{\text{Speed of rope in feet per minute}}{\text{Circumference of driving-wheel}} = \text{revolutions per minute of driving-wheel}$$

$$\frac{\text{Revolutions of engine per minute}}{\text{revolutions of driving-wheel per minute}} = \text{gearing}$$

If a two-cylindered engine is required, calculate in the same way as is given above for Main and Tail.

Examples.—Tonnage per hour = 60. Length of plane = 2000 yards. Average effective steam pressure on piston = 45 lbs. Average gradient against the load = 1 in 10. Weight of full tub = 15 cwt. Weight of empty tub = 5 cwt. Average speed of rope, 1½ mile per hour = 2640 yards per hour.

$$\frac{60\ \text{tons} \times 2000 \times 3}{2640 \times 3} = 45{\cdot}5 \text{ tons of } \textit{coal} \text{ on the rope} = 101920 \text{ lbs.}$$

$$\frac{45{\cdot}5\ \text{tons} \times 20\ \text{cwt.}}{10\ \text{cwt.}} = 91 \text{ full tubs on the rope. Also 91 empties}$$

$= 182$ tubs on the rope

Tubs and rope going inbye balance tubs and rope going outbye.

$$182 \text{ tubs @ 5 cwt.}, \frac{182 \times 5}{20} = 45{\cdot}5 \text{ tons} = 101920 \text{ lbs.}$$

$$\text{Gravity of coal} = \frac{101920}{10} = 10192{\cdot}0$$

$$\text{Friction of tubs} = \frac{101920}{50} = 2038{\cdot}4$$

$$\text{Friction of rope} = \frac{24000}{12} = 2000{\cdot}0$$

14230·4 total load on the driving-wheel (A)

Say a $3\frac{1}{2}$-inch circumference rope, of crucible steel; this weighs 12 lbs. per fathom.

$$2000 \text{ fathoms} \times 12 = 24000 \text{ lbs.}$$

$$\frac{24000}{12} = 2000 \text{ lbs. friction}$$

Actual Size of Rope required.—Factor of safety = 6.

$$\frac{14230{\cdot}4 \text{ (A)}}{2240} = 6{\cdot}35 \text{ tons}$$

$$6{\cdot}35 \times 6 = 38{\cdot}10 \text{ tons breaking-load}$$

By rule on p. 423, a crucible steel rope has a breaking-strain = $3C^2$.

$$3C^2 = 38{\cdot}10$$

$$\therefore C = \sqrt{\frac{38{\cdot}10}{3}} = \sqrt{12{\cdot}7} = 3{\cdot}56 \text{ inches circumference}$$

$$\frac{2640 \text{ yards} \times 3}{60} = 132 \text{ feet per minute speed of rope.}$$ Piston speed, 250 feet per minute

$$\frac{132 \times 14230 \text{ (A)}}{250} = 7513{\cdot}4 \text{ lbs. total pressure required upon the piston (B)}$$

Single Engine.—

$$\frac{7513{\cdot}4 \text{ (B)}}{45 \text{ lbs.}} = 166{\cdot}96, \text{ theoretical area of a single cylinder}$$

Add 50 per cent. for contingencies, such as extra friction on curves and rollers, extra weight such as dirt, friction of engine and driving-wheels, fall in steam pressure, etc.

166·96
83·48
250·44 = area of cylinder

$$\sqrt{\frac{250{\cdot}44}{0{\cdot}7854}} = 17{\cdot}9, \text{ say 18-inch cylinder}$$

Stroke = 18 inches × 2 = 36 inches = 3 feet

$$\text{Revolutions of engine} = \frac{250}{3 \times 2} = 41{\cdot}6$$

Say driving-wheel = 6 feet diameter = 18·8 feet circumference.

$$\frac{132}{18{\cdot}8} = 7 \text{ revolutions per minute of driving-wheel}$$

$$\text{Gearing} = \frac{41{\cdot}6}{7} = 5{\cdot}94, \text{ say 6 to 1}$$

Coupled Engines.—

$$\frac{250{\cdot}44}{2} = 125{\cdot}22 \text{ square inches area of each of the two cylinders}$$

$$\sqrt{\frac{125{\cdot}22}{0{\cdot}7854}} = 12{\cdot}62 \text{ inches diameter, say 13 inches}$$

$$\text{Stroke} = 13 \times 2 = 26 \text{ inches} = 2\ 17 \text{ feet}$$

$$\frac{250}{2 \cdot 17 \times 2} = 57 \cdot 6 \text{ revolutions of engine per minute}$$

Driving-wheel, say, 6 feet diameter.

$$\text{Gearing} = \frac{57 \cdot 6}{7} = 8 \text{ to } 1$$

(As 250 feet of piston speed is not fast for an engine of first-class design and make, the proportions may be altered if desired, so as to give smaller engines a higher gearing, or else the rope may be driven at a greater speed, if such increased speed is suitable to the conditions of haulage.)

Pumps.—*Double-acting.*—Assume pump speed at 100 feet per minute.

$$\frac{\text{Gallons per minute}}{\text{pump speed in feet per minute}} = \text{gallons delivered per foot of stroke if pump is double acting}$$

There are 277·25 cubic inches of water in a gallon

Gallons per foot of stroke × 277·25 = cubic inches of water

$$\frac{\text{Cubic inches of water delivered per foot stroke}}{12} = \text{effective area of ram in square inches}$$

Effective area of ram + area of piston-rod = area of ram

Gallons per minute × 10 × head in feet = foot-pounds of water lifted per minute

If the engine is a direct-acting beam or bull engine, add $\frac{1}{4}$ for friction.
" " vertical, with flywheel " $\frac{1}{3}$ "
" " horizontal " " $\frac{1}{2}$ "
" " geared horizontal " " $\frac{2}{3}$ "

Single Direct-acting Engine.—

$$\frac{\text{Foot-pounds of water lifted per minute + allowance for friction}}{\text{piston speed (100)} \times \text{steam pressure}}$$
$$= \text{area of steam cylinder in square inches}$$

Make stroke = diameter of cylinder × 2. For very large engines the stroke is rather less than double the diameter.

Double Direct-acting Engine, with two double-acting rams.—

$$\text{Effective area of each ram} = \frac{\text{effective area of ram for single pump}}{2}$$
+ area of piston-rod

$$\frac{\text{Foot-pounds of water lifted per minute + allowance for friction}}{\text{piston speed (100)} \times \text{steam pressure} \times 2}$$
= area of steam cylinder

Make stroke = $2d$.

Single-geared Engine.—Piston speed of engine = 250 feet. Piston speed of pump = 100 feet.

$$\text{Effective area of each ram} = \frac{\text{effective area of single ram}}{\text{number of rams}}$$

$$\frac{\text{Ft.-lbs. of water lifted per min.} + \text{allowance for friction}}{\text{piston speed (250)} \times \text{steam pressure} \times \text{gearing}} = \text{area of steam cylinder}$$

Make stroke = $2d$.

Double-geared Engine.—Piston speed of engine = 250 feet. Piston speed of pump = 100 feet.

$$\text{Effective area of each ram} = \frac{\text{effective area of single ram}}{\text{number of rams}}$$

$$\frac{\text{Foot-pounds of water lifted per minute} + \text{allowance for friction}}{\text{piston speed} \times \text{steam pressure} \times \text{gearing} \times 2}$$

= area of each steam cylinder in square inches

Stroke of engine = $2d$

$$\text{Revolutions of engine} = \frac{250}{\text{stroke} \times 2}$$

$$\text{,,} \quad \text{,,} \quad \text{pump} = \frac{\text{revolutions of engine}}{\text{gearing}}$$

Examples of Pumps.—*Double-acting.*—Quantity of water = 34200 + 5 per cent. allowance for slip = 36,000 gallons per hour, to be forced to a height (including friction of water in pipes) = 400 feet. Effective steam pressure upon the piston = 45 lbs. Pump speed = 100 feet per minute. (N.B.—This is about an average speed, but is too fast for some classes of engine.)

$$\frac{36000}{60} = 600 \text{ gallons per minute}$$

$$\frac{600}{100} = 6 \text{ gallons per foot of stroke}$$

$6 \times 277{\cdot}25 = 1663{\cdot}50$ cubic inches of water delivered per foot of stroke

$$\frac{1663{\cdot}5}{12} = 138{\cdot}62 \text{ square inches, effective area of ram}$$

Effective area of ram + area of piston-rod = area of ram (double-acting)

$138{\cdot}62 + 18 = 156{\cdot}62$ = about 14 in. diam. (stroke same as engine)

600 galls. $\times$ 10 $\times$ 400 ft. = 2400000 ft.-lbs. of work in water per min.

Single Horizontal Flywheel Engine.—

$$\frac{2400000 + 1200000}{100 \times 45} = \frac{3600000}{100 \times 45} = 800 \text{ square inches, area of steam cylinder}$$

$$\sqrt{\frac{800}{0{\cdot}7854}} = 31{\cdot}9 \text{ in. diam. stroke} = 31{\cdot}9 \text{ in.} \times 2 = 63{\cdot}8 \text{ in.} = 5 \text{ ft. } 4 \text{ in.}$$

Double Horizontal Flywheel Engine; with two double-acting rams.

$$\frac{2400000 + 1200000}{100 \times 45 \times 2} = \frac{3600000}{100 \times 45 \times 2} = 400 \text{ sq. in., area of steam cylinder}$$

$$\sqrt{\frac{400}{0{\cdot}7854}} = 22{\cdot}6 \text{ inches diameter}$$

22·6 inches × 2 = 45·2 inches = say 3 feet 9 inches stroke
Two cylinders each 23 inches diameter, 3 feet 9 inches stroke

$$\text{Area of rams} = \frac{138·62}{2} = 69·31 \text{ square inches}$$

69·31 + 9 (area piston-rod) = 78·31, area of each ram

$\sqrt{\frac{78·31}{0·7854}}$ = 10 inches diameter (two 10-inch rams, each double-acting; stroke same as engine)

Single-geared Engine.—3 to 1, say. Piston speed of engine, 250. Piston speed of pump, 100. Foot-pounds of work in water + allowance for friction = 2400000 + 1600000 = 4000000.

$$\text{Area of steam cylinder} = \frac{4000000}{250 \times 45 \times 3} = 118·5 \text{ square inches}$$

$$\sqrt{\frac{118·5}{0·7854}} = 12·3 \text{ inches diameter, say 13 inches}$$

13 × 2 = 26 inches stroke = 2 feet 2 inches
Diameter of ram = (138·62 + 18 = 156·62 area) = 14 inches

$$\text{Revolutions of engine} = \frac{250}{2·17 \times 2} = 57·6$$

$$\text{Revolutions of ram-crank} = \frac{57·6}{3} = 19·2$$

$$\text{Stroke of ram} = \frac{100}{19·2 \times 2} = 2·6 \text{ feet, say 2 feet 8 inches}$$

One double-acting ram, 14 inches diameter, 2 feet 8 inches stroke.

Double-geared Engine.—3 to 1 say, with two rams.

$$\frac{4000000}{250 \times 45 \text{ lbs.} \times 3 \times 2} = 59·25 \text{ square inches, area of each steam cylinder}$$

$$\sqrt{\frac{59·25}{0·7854}} = 8·7 \text{ inches diameter}$$

Stroke = 8·7 × 2 = 17·4 = 1 foot $5\frac{1}{2}$ inches

$$\text{Revolutions of engines} = \frac{250}{1·45 \times 2} = 86$$

$$\text{Revolutions of ram-cranks } \frac{86}{3} = 28·66$$

$$\text{Stroke of rams} = \frac{100}{28·66 \times 2} = 1·74 = 1 \text{ foot 9 inches}$$

$$\text{Area of rams} = \frac{156·62}{2} = 78·31 \text{ square inches} = 10 \text{ inches diameter}$$

Say two cylinders, 9 inches diameter, 18 inches stroke.

Two rams, each 10 inches diameter, and each double-acting, with 1 foot 9 inches stroke.

Single-acting Engine.—Pump speed, 100 feet = 50 feet effective one way.

	2400000 foot-pounds work in water lifted per minute
Add one-fourth =	600000 for friction
	3000000 foot-pounds of work in engine per minute

$$\frac{3000000}{\frac{100}{2} \times 45} = 1333 \text{ square inches, effective area of piston}$$

33 square inches, area of piston-rod

1366 area of steam cylinder

$$\sqrt{\frac{1366}{0{\cdot}7854}} = 41{\cdot}7, \text{ say } 42 \text{ inches diameter}$$

Stroke = 42 × 2 = 84 inches = 7 feet

Single-acting Pump.—

$$\frac{600 \text{ gallons a minute}}{50 \text{ feet effective stroke}} = 12 \text{ gallons per foot of stroke}$$

12 × 277·25 = 3327 cubic inches of water per foot of stroke

$$\frac{3327}{12} = 277{\cdot}25 = \text{square inches, effective area of ram}$$

Diameter of ram is 18·8, say 19 inches, and the stroke 7 feet.

Fan Engine.—

Q = quantity of air in cubic feet per minute.
WG = water-gauge in inches.
d = diameter of engine cylinder in inches.
Q × WG × 5·2 = foot-pounds in the air per minute.

$$\frac{\text{Foot-pounds in the air per minute}}{\text{revolutions of engine per minute} \times \text{efficiency (say 0·5 or other fraction)}}$$

= foot-pounds required in the engine per revolution

Make stroke of engine = $1\frac{1}{2}d$.

Piston-travel in inches one revolution of engine = $1\frac{1}{2}d \times 2 = 3d$

then $0{\cdot}7854d^2 \times \text{steam pressure} \times \frac{3d}{12} = \frac{0{\cdot}7854d^3 \times 3 \times \text{steam pressure}}{12}$

= foot-pounds in engine per revolution

$$\therefore \frac{\text{number of foot-pounds as found above per revolution} \times 12}{0{\cdot}7854 \times 3 \times \text{steam pressure}} = d^3$$

The stroke = $1\frac{1}{2}$ diameter of cylinder

Example.—

Q = quantity of air in cubic feet per minute = 100000
WG = water-gauge in inches = 3.
d = diameter of engine cylinder in inches.
s = stroke of engine in inches.
steam pressure, 45 lbs.
efficiency of fan, say 50 per cent., or 0·5.
revolutions of engine per minute to produce the required quantity and WG, say 60.

Q × WG × 5·2 = 100000 × 3 × 5·2 = 1560000 foot-pounds in the air

$$\frac{1560000}{60 \times 0{\cdot}5} = 52000 \text{ foot-pounds in the engine per revolution}$$

Make $s = 1\frac{1}{2}d$.

Piston-travel in feet for one revolution of engine = $\frac{1\frac{1}{2}d \times 2}{12} = \frac{3d}{12}$

and by rule—

$$\frac{0{\cdot}7854d^3 \times 45 \text{ lbs.} \times 3}{12} = 52000 \text{ foot-pounds}$$

$$\therefore \frac{52000 \times 12}{0{\cdot}7854 \times 45 \text{ lbs.} \times 3} = d^3 = 5885$$

$$\sqrt[3]{5885} = 18{\cdot}05, \text{ say } 18 \text{ inches cylinder}$$

$$\text{Stroke} = 18 \text{ inches} \times 1\tfrac{1}{2} = 27 \text{ inches} = 2 \text{ feet } 3 \text{ inches}$$

Single engine, 18 inches diameter, 2 feet 3 inches stroke.

Strength of Round Hemp Ropes.—*Factor of safety* = 10. This high factor is for severe regular work; for occasional loads, a factor of 6 is sufficient.

C = circumference in inches.
B = breaking-strain in tons.
$B = 0{\cdot}25\ C^2$.
$W = 0{\cdot}25\ C^2$.
W = weight per fathom in pounds.
b = safe working-load in tons (factor of safety = 10).

$$b = 0{\cdot}025C^2$$

$$C = \sqrt{\frac{B}{0{\cdot}25}} \qquad C = \sqrt{\frac{b}{0{\cdot}025}}$$

Example.—Circumference of rope = 4 inches = C.

$B = 0{\cdot}25 \times 4^2 = 4$. Breaking-load is 4 tons.
$W = 0{\cdot}25 \times 4^2 = 4$. Weight is 4 lbs. per fathom.
$b = 0{\cdot}025 \times 4^2 = 0{\cdot}4$ Working-load is 0·4 ton = 8 cwt.

If the breaking-load is 8 tons, then $C = \sqrt{\frac{8}{0{\cdot}25}} = \sqrt{32} = 5{\cdot}65$; the circumference of rope should be $5\frac{5}{8}$ inches.

If the working-load is 2 tons, then $C = \sqrt{\frac{2}{0{\cdot}025}} = \sqrt{80} = 8{\cdot}94$; the circumference of the rope should be 9 inches.

Strength and Weight of Chains.—

d = diameter of iron in sixteenths of an inch.
W = breaking-load in tons (chains first-class quality).
w = safe load " " "
P = weight of chain in pounds per length of 10 yards.

$$W = \frac{d^2}{10} \qquad P = d^2$$

For occasional loads, $w = \frac{d^2}{60}$

For regular and severe use (as in cage chains), $w = \frac{d^2}{100}$

$$d = \sqrt{10W}, \quad d = \sqrt{60w}, \quad \text{or } d = \sqrt{100w}$$

Examples.—1-inch chain, $d = 16$.

$$W = \frac{16^2}{10} = \frac{256}{10} = 25{\cdot}6 \text{ tons}$$

$$w = \frac{16^2}{60} = \frac{256}{60} = 4{\cdot}26 \text{ tons for occasional load}$$

$$w = \frac{16^2}{100} = \frac{256}{100} = 2{\cdot}56 \text{ tons for regular and severe use}$$

$\frac{1}{2}$-inch chain, $d = 8$.

$$W = \frac{8^2}{10} = \frac{64}{10} = 6{\cdot}4 \text{ tons breaking-load}$$

$$w = \frac{64}{60} = \text{say 1 ton for occasional load}$$

$$w = \frac{64}{100} = 0{\cdot}64 \text{ ton for regular and severe use}$$

For an occasional load of 2 tons, $d = \sqrt{60 \times 2} = \sqrt{120} = 10{\cdot}95$, say 11; diameter of iron of chain $= \frac{11}{16}$ inch.

For a regular load of 2 tons, $d = \sqrt{100 \times 2} = \sqrt{200} = 14{\cdot}14$, or iron of chain $= \frac{14}{16}$ inch.

Weight of $\frac{3}{4}$ inch ($\frac{12}{16}$) chain—

$$P = d^2 = 12^2 = 144$$

A 10-yard length weighs 144 lbs.

STATUTORY RULES AND ORDERS, 1899.[1]

No. 569.

MINES.

Coal Mines.

THE EXPLOSIVES IN COAL MINES ORDER OF THE 24TH JULY, 1899.

Order made by the Secretary of State for the Home Department under Section 6 of the Ccal Mines Regulation Act, 1896.

WHEREAS by Section 6 of the Coal Mines Regulation Act, 1896, it is enacted that a Secretary of State, on being satisfied that any explosive is, or is likely to become, dangerous, may by Order prohibit the use thereof in any mine or in any class of mines either absolutely or subject to conditions

I hereby, in pursuance of the power conferred on me by the aforesaid section, make the following Order :—

1. (1) In all coal mines in which inflammable gas has been found within the previous three months in such quantity as to be indicative of danger, the use of any explosive, other than a permitted explosive as hereinafter defined, is absolutely prohibited in the seam or seams in which the gas has been found.

(2) In all coal mines which are not naturally wet throughout, the use of any explosive, other than a permitted explosive as hereinafter defined, is absolutely prohibited in all roads, and in every dry and dusty part of the mine.

2. In all such coal mines or parts thereof as aforesaid, the use of permitted explosives is prohibited unless the following conditions are observed :—

(*a*) Every charge of the explosive shall be placed in a properly drilled shot hole, and shall have sufficient stemming :

[1] Reprinted by permission of the Controller of Her Majesty's Stationery Office.

(*b*) Every charge shall be fired by an efficient electrical apparatus, or by some other means equally secure against the ignition of inflammable gas or coal dust:

(*c*) Every charge shall be fired by a competent person appointed in writing for this duty by the owner, agent, or manager of the mine, and not being a person whose wages depend on the amount of mineral to be gotten:

(*d*) Each explosive shall be used in the manner and subject to the conditions prescribed in the Schedule hereto:

Provided that nothing in this Order shall prohibit the use of a safety fuze in any mine in which inflammable gas has not been found within the previous three months in such quantity as to be indicative of danger.

3. In every coal mine the use of any explosive is prohibited in the main haulage roads and in the intakes unless all workmen have been removed from the seam in which the shot is to be fired, and from all seams communicating with the shaft on the same level, except the men engaged in firing the shot, and in addition such other persons, not exceeding ten in number, as are necessarily employed in attending to the ventilating furnaces, steam boilers, engines, machinery, winding apparatus, signals, or horses, or inspecting the mine; or unless a permitted explosive is used, and every part of the roof, floor, and sides of the main haulage road or intake, within a distance of 20 yards from the place where it is used, is, at the time of firing, thoroughly wet, either naturally or from the application of water thereto.

This section shall not apply to such portions of the main haulage roads and intakes as are within 100 yards of the coal face.

This section shall not authorize the use of any explosive in any case where the use of such explosive is prohibited by Section 1 or 2 of this Order.

4. On and after the 1st day of October, 1899, no detonator shall be used in any mine unless the following conditions are observed:—

(*a*) Detonators shall be under the control of the owner, agent, or manager of the mine, or some person specially appointed in writing by the owner, agent, or manager for the purpose, and shall be issued only to shot firers or other persons specially authorized by the owner, agent, or manager, in writing.

(*b*) Shot firers and other authorized persons shall keep all detonators issued to them until about to be used in a securely locked case or box separate from any other explosive.

5. This Order shall not apply to mines of clay, or stratified or nodular ironstone, nor shall it apply to shafts in course of being sunk from the surface, or deepened, or to drifts and other outlets being driven from the surface, if such shafts, drifts, or outlets are not ventilated by return air.

Where a mine contains several separate seams, this Order shall apply to each seam as if it were a separate mine.

6. In this Order the term "permitted explosives" means such explosives as are named and defined in the Schedule hereto: provided

that where the composition, quality, or character of any explosive is defined in such Schedule, any article alleged to be such explosive which differs therefrom in composition, quality, or character, whether by reason of deterioration or otherwise, shall not be deemed to be the explosives so defined; provided further that an owner, agent, or manager shall not be responsible for the composition, quality, or character of an explosive, if he shows that he has in good faith obtained a written certificate from the maker of the explosive that it complies with the terms of the Schedule, and that he has taken all reasonable means to prevent deterioration of the explosive while stored.

The term "road" includes all roads of any description extending from the shaft or outlet to within 10 yards of the coal face.

The term "main haulage road" means a road which has been, or for the time being is, in use for moving trams by gravity or by steam or other mechanical power.

7. This Order shall come into force on the 1st day of August, 1899, from which date the Explosives in Coal Mines Orders of the 11th July, 1898, and 23rd December, 1898, are revoked.

8. This Order may be cited as the Explosives in Coal Mines Order of the 24th July, 1899.

M. W. RIDLEY,
One of Her Majesty's Principal
Secretaries of State.

Home Office, Whitehall,
24th July, 1899.

Schedule.

LIST OF PERMITTED EXPLOSIVES.[1]

Ammonite, consisting in every 100 parts by weight of the finished explosive of not more than 89 parts and not less than 87 parts of nitrate of ammonium, with not more than 13 parts and not less than 11 parts of thoroughly purified di-nitro-naphthalene, and with no other ingredient; the whole being uniformly incorporated;

Provided—

(1) That the explosive shall be used only when contained in a case of lead and tin alloy thoroughly waterproofed with pure paraffin wax;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6½ (*i.e.* the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 19 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium); and

[1] This list is subject to revision in accordance with the results of experiments made from time to time in the Government Testing Station at Woolwich.

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 6½ detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Amvis, consisting in every 100 parts by weight of the finished explosive of not more than 92 parts and not less than 89 parts of nitrate of ammonium, with not more than 6 parts and not less than 4 parts of wood-meal, and with not more than 6 parts and not less than 4 parts of thoroughly purified di-nitro-benzol and chlorinated naphthalene, and with no other ingredient, provided that the chlorine does not exceed 1 per cent. by weight of the finished explosive;

Provided—

(1) That the explosive shall be used only when contained in a case of stout paper thoroughly waterproofed with ceresine;

(2) That the explosive shall be used only with a special detonator or electric detonator fuze containing not less than 15 grains of a composition consisting in every 100 parts by weight of 95 parts of fulminate of mercury and 5 parts of chlorate of potassium; and

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with a special detonator"; and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Bellite No. 1, consisting in every 100 parts by weight of the finished explosive of not more than 85 parts and not less than 82 parts of nitrate of ammonium, with not more than 18 parts and not less than 15 parts of thoroughly purified di-nitro-benzol, and with no other ingredient; the whole being uniformly incorporated;

Provided—

(1) That the explosive shall be used only when contained in a case of linen paper thoroughly waterproofed with a mixture of carnauba and paraffin waxes;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 7 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 23 grains of a composition consisting in every 100 parts by

weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium) ; and

(3) That, in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives" ; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used with not less than No. 7 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Bellite No. 3, consisting in every 100 parts by weight of the finished explosive of not more than 95 parts and not less than 92 parts of nitrate of ammonium, with not more than 8 parts and not less than 5 parts of thoroughly purified di-nitro-benzol, and with no other ingredient ; the whole being uniformly incorporated ;

Provided—

(1) That the explosive shall be used only when contained in a case of linen paper thoroughly waterproofed with a mixture of carnauba and paraffin waxes ;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium) ; and

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Benedite, consisting in every 100 parts by weight of the finished explosive of not more than 95 parts and not less than 92 parts of neutral nitrate of ammonium, with not more than 7 parts and not less than 5 parts of colophony which does not melt below 200 degrees Fahrenheit, and with no other ingredient ;

Provided—

(1) That the explosive shall be used only when contained in a paper case thoroughly waterproofed with ceresine, linseed oil and resin, and with or without the addition of carbonate or bi-carbonate of sodium, or alum ;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 8 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 30·9 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium); and

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 8 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

British Gelignite, consisting in every 100 parts by weight of the finished explosive of not more than 62 parts and not less than 58 parts of thoroughly purified nitro-glycerine, with not more than 5 parts and not less than 3 parts of nitro-cotton, carefully washed and purified, and not more than 31 parts and not less than 26 parts of nitrate of potassium, and not more than 9 parts and not less than 6 parts of wood-meal, and with no other ingredient; the whole being uniformly incorporated and of such character and consistency as not to be liable to exudation;

Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of parchment paper.

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium).

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed in a safe and suitable manner before use.

Bulldog Brand Gunpowder, consisting in every 100 parts by weight of the finished explosive of not less than 83·5 parts and not more than 86·5 parts of pure saltpetre, with not less than 13 parts or more than 14 parts of charcoal, and not less than one part and not more than 2 parts of pure distilled sulphur, and with not less than one part and 2·5 parts of moisture, and with no other ingredient, the whole being thoroughly well incorporated, and to be of such strength that five parts, when exploded in a lead cylinder as used at the Home Office Testing Station, will give a result not inferior to that obtained by four parts of R.F.G.$_2$ gunpowder, and to be in the form of grains of a size to pass through a sieve of 10 meshes to the linear inch, and to be retained on a sieve of 40 meshes to the linear inch;

Provided—

(1) That the explosive may, if so required, be compressed into a pellet of density not exceeding 1·4.

(2) That the explosive, whether in grain or pellet form, shall not be taken into or used in a mine except when contained in a spark-proof brown paper case or cartridge.

(3) That when the saltpetre is washed out, the residue of charcoal, after being dried at 330° Fahrenheit, must lose not less than 22 per cent. of volatile matter other than sulphur when heated to redness in a current of coal gas, and that when the said charcoal is burned in air, the residue of mineral matter shall be not more than 5 per cent.

(4) That the explosive when in grain shall be in all other respects similar to the sample submitted for test on the 14th March, 1899, and when in pellet form, to that submitted on the 23rd March, 1899.

(5) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Carbo-gelatine, consisting in every 100 parts by weight of the finished explosive of not more than 40 parts and not less than 37 parts of a mixture of carefully washed nitro-cotton and thoroughly purified nitro-glycerine, with not more than 51 parts and not less than 48 parts of nitrate of potassium, and with not more than 12 parts and not less than 9 parts of a mixture of wood-meal and charcoal, provided that the charcoal shall not exceed 3 parts by weight in every 100 parts by weight of the finished explosive, and not more than 2 parts of carbonate of magnesium and with no other ingredient; the whole to be thoroughly mixed or incorporated so as not to be liable to liquefaction or exudation;

Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed parchment paper wrapper.

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 6 detonator," and marked with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients;

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed in a safe and suitable manner before use.

Carbonite, consisting in every 100 parts by weight of the finished explosive of not more than 27 parts and not less than 25 parts of thoroughly purified nitro-glycerine, with not more than 36 parts and not less than 30 parts of nitrate of barium and nitrate of potassium or either of them, and with not more than 43 parts and not less than 40 parts of wood-meal, with or without not more than half a part of sulphuretted benzol and not more than half a part of carbonate of sodium and carbonate of calcium or either of them, and with no other ingredient; the whole being uniformly incorporated and of such character and consistency as not to be liable to exudation;

Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of parchment paper;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 6

detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed in a safe and suitable manner before use.

Dahmenite A, consisting in every 100 parts by weight of the finished explosive of not more than 93·5 parts and not less than 91 parts of nitrate of ammonium, with not more than 6·5 parts and not less than 4 parts of naphthalene, and with not more than 2·5 parts and not less than one part of bichromate of potassium, and with no other ingredient;

Provided—

(1) That the explosive shall be used only when contained in paper wrappers, waterproofed with paraffin wax and resin;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 7 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 23 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium); and

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 7 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Earthquake Powder, consisting in every 100 parts of the finished explosive of not more than 81 parts and not less than 78 parts of pure saltpetre, with not more than 22 parts and not less than 19 parts of charcoal, and with or without the addition of $\frac{1}{8}$ per cent. of pure sulphur, and with no other ingredient, the whole being thoroughly well incorporated, and to be of such strength as, when exploded in a lead cylinder as used at the Home Office Testing Station, will give a result not inferior to that obtained with an equal weight of $R.F.G._2$ gunpowder, and to be in the form of grains of a size to pass through a sieve of 11 meshes to the linear inch, and to be retained by a sieve of 40 meshes to the linear inch.

Provided—

(1) That the gunpowder shall not be taken into or used in a mine except when contained in a parchment paper case or wrapper.

(2) That when the saltpetre is washed out, the residue of charcoal dried at 230° Fahr. must lose not less than 56 per cent. by weight of volatile matter when heated to redness in a current of coal gas, and that when the said charcoal is burned in

air the residue of mineral matter or ash shall not exceed 1·5 per cent. by weight.

(3) That the explosive shall be in all other respects similar to the sample submitted for test on the 15th August, 1898.

(4) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Electronite No. 2, consisting in every 100 parts by weight of the finished explosive of not more than 96 parts and not less than 94 parts of neutral nitrate of ammonium, with not more than 6 parts and not less than 4 parts of wood-meal and starch, and with no other ingredient;

Provided—

(1) That the explosive shall be used only when contained in a waterproof metal case made of an alloy of lead and tin, or in a paper wrapper waterproofed with ceresine;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium); and

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Electronite No. 3, consisting in every 100 parts by weight of the finished explosive of not more than 75 parts and not less than 70 parts of neutral nitrate of ammonium; with not more than 21 parts and not less than 16 parts of nitrate of barium; and with not more than 9 parts and not less than 6 parts of a mixture of wood-meal, slightly charred, starch, and pure pine resin which does not melt below 200° Fahr., and with no other ingredient, and to be in the form of grains of a size to pass through a sieve of 12 meshes to the linear inch;

Provided—

(1) That the explosive shall be used only when contained in a paper wrapper waterproofed with ceresine, or in a case of lead thoroughly waterproofed;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 7 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 23 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That the explosive shall be in all other respects similar to the sample submitted for test on the 8th June, 1899; and

(4) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 7 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture and the nature and proportion of the ingredients.

Elephant Brand Gunpowder, consisting in every 100 parts by weight of the finished explosive of not more than 76 parts and not less than 74 parts of pure saltpetre, with not more than 15½ parts and not less than 14½ parts of charcoal, and not more than 11 parts and not less than 9 parts of pure distilled sulphur, and with no other ingredient, the whole being thoroughly well incorporated, and to be of such strength as, when exploded in a lead cylinder as used at the Home Office Testing Station, will give a result not inferior to that obtained with an equal weight of R.F.G.$_2$ gunpowder, and to be in the form of grains of a size to pass through a sieve of 11 meshes to the linear inch;

Provided—

(1) That the gunpowder shall not be taken into or used in a mine except when contained, together with neutral oxalate of ammonium in the proportion of 1 part by weight of oxalate of ammonium to 2 parts by weight of gunpowder, in a spark-proof brown paper case or cartridge (Elephant Brand), in which there shall intervene between the gunpowder and the oxalate of ammonium a diaphragm of such strength and character as will effectually prevent any admixture of the two;

(2) That there shall not be taken into or used in a mine any case or cartridge containing more than 9 ozs. of the said gunpowder; that every case or cartridge shall be inserted intact in the hole, and that not more than one case or cartridge at a time shall be inserted;

(3) That no shot with the said gunpowder shall be fired unless properly stemmed with an amount of stemming not less than would be sufficient for a charge of 9 ozs. of ordinary gunpowder;

(4) That the cases or cartridges shall be packed in thoroughly waterproofed wrappers, bags, or other receptacles, each containing not more than 5 lbs. of gunpowder; and

(5) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each cartridge shall be clearly marked with the words "Permitted Explosive," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Elephant Brand Gunpowder No. 2, consisting in every 100 parts by weight of the finished explosive of not more than 76 parts and not less than 74 parts of pure saltpetre, with not more than 15½ parts and not less than 14½ parts of charcoal, and not more than 11 parts and not less than 9 parts of pure distilled sulphur, and with no other ingredient, the whole to be thoroughly well incorporated, and to be of such strength as, when exploded in a lead cylinder as used at the Home Office Testing Station, will give a result not inferior to that obtained with an equal weight of $R.F.G._2$ gunpowder, and to be in the form of grains of a size to pass through a sieve of 11 meshes to the linear inch;

Provided—

(1) That the gunpowder shall not be taken into or used in a mine except when contained, together with pure bi-carbonate of sodium in the proportion of 1 part by weight of bi-carbonate of sodium to 2 parts by weight of gunpowder, in a spark-proof brown paper case or cartridge (Elephant Brand), in which there shall intervene between the gunpowder and the bi-carbonate of sodium a diaphragm of such strength and character as will effectually prevent any admixture of the two;

(2) That there shall not be taken into or used in a mine any case or cartridge containing more than 9 ozs. of the said gunpowder; that every case or cartridge shall be inserted intact in the hole, and that not more than one case or cartridge at a time shall be inserted;

(3) That no shot with the said gunpowder shall be fired unless properly stemmed with an amount of stemming not less than would be sufficient for a charge of 9 ozs. of ordinary gunpowder;

(4) That the cases or cartridges shall be packed in thoroughly waterproofed wrappers, bags, or other receptacles, each containing not more than 5 lbs. of gunpowder; and

(5) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and,

further, that each cartridge shall be clearly marked with the words " Permitted Explosive," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Faversham Powder, consisting in every 100 parts by weight of the finished explosive of not more than 87 parts and not less than 83 parts of nitrate of ammonium, with not more than 14 parts and not less than 9 parts of thoroughly purified di-nitro-benzol, with not more than 2 parts and not less than 1 part of chloride of ammonium and not more than 3 parts and not less than 2 parts of chloride of sodium, and with no other ingredient ; the whole being uniformly incorporated ;

Provided—

(1) That the explosive shall be used only when contained in a case of paper thoroughly waterproofed with paraffin wax, and with or without a lead nozzle ;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as 6½ (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 19 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium) ;

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words " As defined in the List of Permitted Explosives " ; and, further, that each inner package shall be clearly marked with the words " Permitted Explosive, to be used only with not less than 6½ detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Kynite, consisting in every 100 parts by weight of the finished explosive of not more than 27 parts and not less than 25 parts of thoroughly purified nitro-glycerine, with not more than 36 parts and not less than 30 parts of nitrate of barium, and not more than 43 parts and not less than 40 parts of wood-meal, and with not more than half a part of carbonate of sodium, and with no other ingredient ; the whole being uniformly incorporated and of such character and consistency as not to be liable to exudation ;

Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of vegetable parchment ;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to

be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed in a safe and suitable manner before use.

Kynoch Gelignite, consisting in every 100 parts by weight of the finished explosive of not more than 63 parts and not less than 54 parts of thoroughly purified nitro-glycerine, with not more than 5 parts and not less than 3 parts of nitro-cotton, carefully washed and purified, and not more than 34 parts and not less than 26 parts of nitrate of potassium, and not more than 9 parts and not less than 6 parts of wood-meal, and with or without not more than one part of chalk, and with no other ingredient; the whole being uniformly incorporated and of such character and consistency as not to be liable to exudation;

Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of parchment paper;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed in a safe and suitable manner before use.

Nahnsen's Gelignite, consisting in every 100 parts by weight of the finished explosive of not more than 63 parts and not less than 54 parts of thoroughly purified nitro-glycerine, with not more than 5 parts and not less than 3 parts of nitro-cotton, carefully washed and purified, not more than 34 parts and not less than 26 parts of nitrate of potassium, and not more than 10 parts and not less than 6 parts of wood-meal, and with or without not more than half a part of chalk, and with no other ingredient; the whole being uniformly incorporated and of such character and consistency as not to be liable to exudation;

Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of parchment paper;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed in a safe and suitable manner before use.

National Gelignite, consisting in every 100 parts by weight of the finished explosive of not more than 64 parts and not less than 56 parts of thoroughly purified nitro-glycerine, with not more than 6 parts and not less than 4 parts of nitro-cotton, carefully washed and purified, and not more than 32 parts and not less than 24 parts of nitrate of potassium, and not more than 9 parts and not less than 5 parts of wood-meal, and with or without not more than half a part of chalk, and with no other ingredient; the whole being uniformly incorporated and of such character and consistency as not to be liable to exudation;

Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of parchment paper;

(2) That the explosive shall be used only with a special detonator or electric detonator fuze containing not less than 15 grains of a composition consisting in every 100 parts by weight of 95 parts of fulminate of mercury and 5 parts of chlorate of potassium;

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with a special detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed in a safe and suitable manner before use.

Nobel Ardeer Powder, consisting in every 100 parts by weight of the finished explosive of not more than 34 parts and not less than 31 parts of thoroughly purified nitro-glycerine, with not more than 14 parts and not less than 11 parts of kieselguhr, with not more than 51 parts and not less than 47 parts of sulphate of magnesium, and with not more than 6 parts and not less than 4 parts of nitrate of potassium, with or without the addition of not more than half a part of carbonate of ammonium and not more than half a part of carbonate of calcium, and with no other ingredient; the whole being uniformly incorporated and of such character and consistency as not to be liable to exudation;

Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of parchment paper;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 3 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 8 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked by the words "Permitted Explosive, to be used only with not less than No. 3 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed in a safe and suitable manner before use.

Nobel Carbonite, consisting in every 100 parts by weight of the finished explosive of not more than 27 parts and not less than 25 parts of thoroughly purified nitro-glycerine, and not more than 36 parts and not less than 30 parts of nitrate of potassium, and nitrate of barium, or either of them, and with not more than 43 parts and not less than 40 parts of wood-meal, with or without not more than half a part of sulphuretted benzol, and not more than half a part of carbonate of sodium and carbonate of calcium, or either of them, and with no other ingredient; the whole being uniformly incorporated and of such character and consistency as not to be liable to exudation;

Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of parchment paper;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed in a safe and suitable manner before use.

Nobel Gelignite, consisting in every 100 parts by weight of the finished explosive, of not more than 63 parts and not less than 54 parts of thoroughly purified nitro-glycerine, with not more than 5 parts and not less than 3 parts of nitro-cotton, carefully washed and purified, not more than 34 parts and not less than 26 parts of nitrate of potassium, and not more than 9 parts and not less than 6 parts of wood-meal, and with or without not more than half a part of chalk, and with no other ingredient; the whole being uniformly incorporated, and of such character and consistency as not to be liable to exudation;

Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of parchment paper;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a

composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed in a safe and suitable manner before use.

Oxalate Blasting Powder, consisting in every 100 parts by weight of the finished explosive of not more than 73 parts and not less than 69 parts of nitrate of potassium, with not more than 15½ parts and not less than 12 parts of charcoal, with not more than 16½ parts and not less than 13½ parts of oxalate of ammonium, and with or without not more than two parts of sulphur, and with no other ingredient; the whole being thoroughly incorporated.

Provided—

(1) That the explosive shall be used only when contained in non-waterproofed papers of (*a*) an alloy of lead and tin or (*b*) asbestos paper.

(2) That in addition to the marking on the outer package, required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives"; and, further, that each inner package shall be clearly marked with the words "Permitted Explosive," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Pembrite, consisting in every 100 parts by weight of the finished explosive of not more than 96 parts and not less than 93 parts of neutral nitrate of ammonium, with not more than 6 parts and not less than 3 parts of vegetable oil of a character approved by the Secretary of State, and with not more than 2 parts and not less than 1 part of sulphur, and with or without not more than 1 part of nitrate of barium, and with no other ingredient; the whole being uniformly incorporated;

Provided—

(1) That the explosive shall be used only when contained in non-waterproofed wrappers of metal-coated paper;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known

as No. 8 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 30·9 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium); and

(3) That in addition to the marking on the outer package required in the case of this explosive by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives;" and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 8 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Rhenish Gelignite, consisting in every 100 parts by weight of the finished explosive of not more than 59 parts and not less than 57 parts of thoroughly purified nitro-glycerine, with not more than 3 parts and not less than 2 parts of nitro-cotton, carefully washed and purified, not more than 31 parts and not less than 28 parts of nitrate of potassium, and not more than 10½ parts and not less than 9 parts of wood-meal, and with no other ingredient; the whole being uniformly incorporated and of such character and consistency as not to be liable to exudation;

Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of parchment paper;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives;" and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed in a safe and suitable manner before use.

Roburite No. 3, consisting in every 100 parts by weight of the finished explosive of not more than 89 parts and not less than 86 parts of nitrate of ammonium, with not more than 13 parts and not less than 9 parts of thoroughly purified dinitro-benzol, with or without not more than two parts of chloro-naphthalene containing of chlorine not more than one part, and with no other ingredient; the whole being uniformly incorporated;

Provided—

(1) That the explosive shall be used only when contained in a case of paper thoroughly waterproofed with ceresine;

(2) That the explosive shall be used only with a special detonator or electric detonator fuze containing not less than 15 grains of a composition consisting in every 100 parts by weight of 95 parts of fulminate of mercury and 5 parts of chlorate of potassium; and

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives;" and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with a special detonator," and also with the name of the explosive, the name of the manufacturer, date of manufacture, and the nature and proportion of the ingredients.

Sun Gelignite, consisting in every 100 parts by weight of the finished explosive of not more than 59 parts and not less than 57 parts of thoroughly purified nitro-glycerine, with not more than 3 parts and not less than 2 parts of nitro-cotton, carefully washed and purified, not more than 31 parts and not less than 25 parts of nitrate of potassium, and not more than 10 parts and not less than 8 parts of wood-meal, and with no other ingredient; the whole being uniformly incorporated and of such character and consistency as not to be liable to exudation;

Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of parchment paper;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives;" and, further, that each inner package shall be clearly marked with the words

"Permitted Explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed in a safe and suitable manner before use.

Westfalite No. 1, consisting in every 100 parts by weight of the finished explosive of not more than 96 parts and not less than 94 parts of neutral nitrate of ammonium, with not more than 6 parts and not less than 4 parts of resin, consisting of pure pine resin which does not melt below a temperature of 200° Fahr., and with no other ingredient; the whole being uniformly incorporated;

Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of paper;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 8 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 30·9 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium); and

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives;" and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 8 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Westfalite No. 2, consisting in every 100 parts by weight of the finished explosive of not more than 92 parts and not less than 90 parts of neutral nitrate of ammonium, with not more than 5 parts and not less than 3 parts of nitrate of potassium, and with not more than 6 parts and not less than 4 parts of resin, consisting of pure pine resin which does not melt below a temperature of 200° Fahr., and with no other ingredient; the whole being uniformly incorporated;

Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of paper;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 8 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 30·9 grains of a composition consisting in every 100 parts by weight of 80

parts of fulminate of mercury and 20 parts of chlorate of potassium); and

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "As defined in the List of Permitted Explosives;" and, further, that each inner package shall be clearly marked with the words "Permitted Explosive, to be used only with not less than No. 8 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

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